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MINING METHODS AND COSTS AT THE MAGMA MINE, SUPERIOR, ARIZ.

BY

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MINING METHODS AND COSTS AT THE MAGMA MINE, SUPERIOR, ARIZ.[1]

By Fred W. Snow[2]

INTRODUCTION

This paper describing the mining practices at the Magma mine, Superior, Ariz., is one of a series being prepared by the Bureau of Mines on mining practices, methods, and costs in the various mining districts in the United States.

At present (1929) practically all the mineral production in the Superior district is from the Magma mine of the Magma Copper Co. The ore is mined for its copper but contains associated silver and gold. Both direct smelting ore and ore of concentrating grade are produced.

The mining methods now in use at the Magma mine were adopted or developed as knowledge was gained of the physical condition of the ore bodies and the inclosing formations; the wall rock is soft and crushed and the ore is relatively hard. In general, existing methods have been adapted to fit local conditions, but some of the mining practices used were originated at this mine.

Most of the ore is mined by a combination of rill stopes and pillar stopes. The pillars are extracted by underhand mining.

ACKNOWLEDGMENTS

The author acknowledges the assistance of William Koerner, general manager, E. G. Dentzer, assistant general manager, Herman Forsnas, mine superintendent, J. F. Flanagan, chief engineer, Darrell Gardner, mine foreman, J. R. Mays, auditor, and C. B. Foraker, safety engineer of the Magma mine, and of E. D. Gardner, supervising engineer, G. E. McElroy, mining engineer, and C. H. Johnson, assistant engineer, U. S. Bureau of Mines, Southwest Experiment Station, Tucson, Ariz.

HISTORY OF THE DISTRICT AND EARLY DEVELOPMENT

The Magma mine is situated in the Pioneer (Superior) mining district, Pinal County, Ariz., about 70 miles southeast of Phoenix and 22 miles west of Miami. The elevation of the mine is 3,500 feet. Superior is served by a 30-

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2 One of the consulting engineers, U. S. Bureau of Mines.

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mile standard-gage railroad which connects with a branch of the Southern Pacific Railroad system at Magma, Ariz.

The discovery in 1875 of the Silver King mine, one of the famous old Arizona silver producers, first attracted attention to the Superior district. The Silver Queen, which later became the Magma, was located in the same year about 3 miles south of the Silver King. In the early days the Queen mine was worked for silver, which was, however, associated with considerable chalcocite. The Magma Copper Co. was organized in 1910 and has acquired by the purchase and location of several groups of claims a large and compact area at Superior.

Previous to 1910 the underground work at the Magma mine consisted of the No. 1 shaft, which had been sunk to the 400-foot level, and a few hundred feet of drifting and crosscutting on the 100, 200, 300, and 400 foot levels, where small isolated bodies of high-grade ore were developed and stoped. They were probably mined by the square-set method, but as the workings were inaccessible at the time the Magma Copper Co. took over the property, the method used is not known.

From 1910 to 1914 the No. 1 shaft was sunk to a depth of 800 feet, and active exploration started on several levels. The first important bodies of ore were discovered on the 600 and 800 foot levels during this period.

An electric power line was completed in 1914 connecting the Magma mine with the power line from the Roosevelt dam by way of the Inspiration mine. Early the same year a two-compartment winze was sunk from the 800 to the 1,000 foot level where a high-grade bornite ore body was discovered. This winze was later enlarged to three compartments and became a part of No. 2 shaft.

A concentrator having a capacity of 150 tons per day was completed in 1914.

All supplies for the district and all ore and concentrates shipped were hauled by team between Superior and Florence, the railroad point 32 miles away. In 1915 a narrow-gage railroad was constructed to serve Superior and was changed over to the present standard-gage road in 1923.

By the end of 1916 No. 2 shaft had been sunk from the 200-foot, the main adit level at that time, to below the 1,600-foot level. A double-drum electric hoist was installed underground on the 200 level. In 1917 a second working shaft to the lower levels became necessary. As the two-compartment No. 1 shaft was in bad condition near the 100 level due to the early stoping operations, it was decided to sink a new three-compartment shaft (No. 3). By the end of 1919 both No. 2 and No. 3 shafts had been sunk to below the 2,000-foot levels; also, the 500 adit had been connected to all shafts, and was used for the main outlet of the mine. At present, the portal of this adit is connected with the various parts of the surface plant by means of a standard-gage railroad system, which in turn is connected with the branch line to Magma.

In 1919 No. 4 shaft was started from the surface to be used as the main outlet for the ventilation system. During the general shut-down of the copper mines in 1921 and 1922, No. 2 shaft was sunk to below the 2,250-foot level. Development work was pushed on the 1,800 and 2,000 foot levels, and a

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large body of ore was blocked out, sufficient to warrant the building of a smelter and increasing the capacity of the concentrator to 750 tons per day. These improvements were completed early in 1924.

For the economical operation of the western part of the mine, No. 5 shaft was begun in 1926 and completed to below the 2,960 level late in 1928. It connects with a short adit on the 500 level. Figure 1 shows the relationship of the shafts and other main workings of the mine.

PRODUCTION

In 1928 the Magma mine produced a total of 263,094 tons of all classes of ore, from which 35,228,810 pounds of copper was made. The ore mines averaged 7.19 per cent copper and contained 3,42 ounces of silver and 0.034 ounces of gold per ton. In 1928 1.703 tons of ore was mined per man-shift underground, excluding men on salaries.

The Annual Reports of the Magma Copper Co. show that from the beginning of the Magma Copper Co.'s operations in 1910 to the end of 1928 the production was 214,000,000 pounds of copper from 1,860,000 tons of ore, in addition to 6,965,000 ounces of silver and 72,000 ounces of silver and 72,000 ounces of gold.

GEOLOGY

The strata at Superior are the same as those at Globe and Ray, as described by Ransome.[3] The basal formation is a pre-Cambrian schist. Overlying the schist is a series of Paleozoic sedimentaries consisting of conglomerates, shales, quartzites, Devonian limestone, lower Carboniferous limestone, and upper Carboniferous limestone. The dip of the formations is from 30 to 35° to the east.

The chief igneous rocks of the district are a large premineral diabase sill which has been intruded into the Paleozoic sediments, and a postmineral dacite flow which covers in many sections the older formations. The dacite overlies the eastern extension of the Magma vein. In addition, dikes of premineral quartz monzonite porphyry intrude boththe sediments and the diabase, and numerous small postmineral dikes of basalt are found.

The principal ore bodies in the district occur as shoots in a strong east and west faultefissure that strikes at right angles to the strike of the beds. The fissure cuts the diabase and the sedimentary formations but not the overlying dacite, which is later than the faulting. Relatively small unimportant carbonate ore bodies are found in some parts of the district on the contact between the upper quartzite and the Devonian limestone.

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3 Ransome, F. L., The Copper Deposits of Ray and Miami, Ariz. U. S. Geol. Survey Prof. Paper 115, 1919, 192 pp. See also Ransome, F. L., Copper Deposits near Superior, Ariz. U. S. Geol. Survey Bull. 540, 1914, p. 139; Short, M. N., and Efflinger, I. A., Ore Deposition and Enrichment at the Magma Mine, Superior, Ariz. Trans. Am. Inst. Min.and Met. Eng., vol. 74, 1926. p. 174.

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Geology of Magma Mine

The structural features having an influence upon the mining of the Magma ore bodies are: (1) The east and west fault which formed the vein; (2) the postmineral faulting at approximately right angles to the strike of the vein (3) the postmineral movement along the vein, probably coincident with the transverse faulting.

The vein strikes nearly due east and west, cutting all formations existing at the time the faulting took place, and dips from 45 to 80° to the north down to the 900 level at No. 2 shaft. At the 900 level the vein turns over, maintaining a constant dip to the lowest levels of the mine of about 80° to the south. The reversal of the dip takes place lower in the western part of the mine than in the eastern part. The vein filling is altered mineralized country rock.

The various contacts in the north and south walls of the vein indicate that there has been a combined horizontal and vertical displacement of approximately 600 feet, the south wall having moved west and down with respect to the north wall.

The north-south fault has a large displacement, and cuts the vein into several blocks in the western part of the mine. This increases exploration costs as well as the cost of mining the ore in these faulted blocks.

The postmineral movement along the vein resulted in the formation of treacherous hanging and foot walls. Considerable gouge and crushed wall rock, with frequent included lenses of ore, occur at most places along the walls of the wider parts of the vein. In places where the vein is narrow (5 to 10 feet) this crushed condition may extend from wall to wall, and cause very heavy ground.

The ores of the Magma mine may be classified into two general types; one in which the predominating gangue is altered partly-silicified diabase or porphyry, and one in which the predominating gangue is quartz. This classification holds true in the primary zone and, to a less extent, in the secondary zone. On account of the silicification, the ore is generally harder to drill and break than the wall rocks.

The ore deposits within the fissure are extensive, both on the strike and dip, and their size and grade have increased with depth. Most of the ore mined to date has come from a series of closely-related ore bodies extending from just below the 400 level down to the deepest levels in the mine. The principal ore shoot on the lower levels of the mine is 1,200 to 1,500 feet long, averaging 20 to 30 feet wide, and as mined runs from 6 to 8 per cent copper. Other ore shoots which may develop into important ore bodies are being prospected.

The Magma ore bodies are valuable chiefly for their copper, but they have important amounts of associated silver and gold. Zinc sulphide ore of good grade is found in a number of places and may prove of value in the future.

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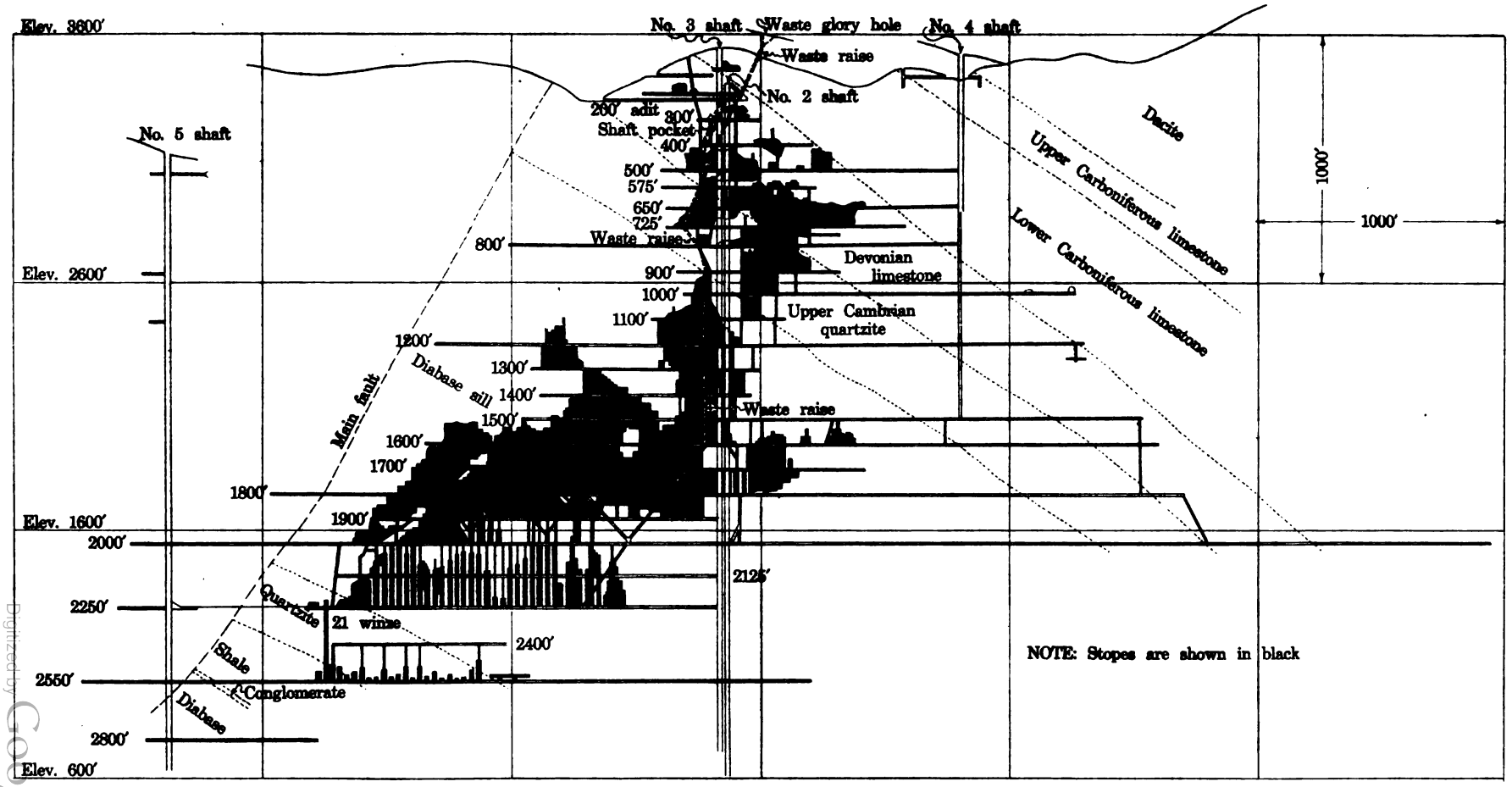

Figure 1.– Longitudinal projection of Magma mine, looking north. The geologic structure shown is that of the south or hanging wall of vein

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The ore mined above the 400 level was largely oxidized, although it was associated with some chalcocite. The important copper minerals in the oxidized zone are malachite, chrysocolla, chalcocite, and some cuprite. A zone of secondary enrichment extends from the 400 level to about the 900, in which the chief copper minerals are chalcocite and some bornite, associated with pyrite. The primary ores below the 900 level contain principally bornite, chalcopyrite, and pyrite. In the deepest levels some primary chalcocite and tetrahedrite are found.

Gold and silver are associated with all classes of ore, which usually carry about one-half ounce of silver to each per cent of copper and 50 to 75 cents a ton in gold.

METHODS OF PROSPECTING AND EXPLORATION

The ore bodies in the Magma vein have been prospected and explored by means of shafts, drifts, crosscuts, winzes, and raises.

In the past a great deal of diamond drilling has been done to obtain data on the formations, rather than to determine ore values. No diamond drilling is being done at present.

METHODS OF SAMPLING AND ESTIMATING TONNAGE AND GRADE

Groove samples are cut with hammer and moil in all crosscuts and drifts in the vein. Crosscuts are sampled waist-high and parallel to the floor. The length of each sample cut depends upon the ore but is in no case longer than 5 feet. The Magma vein has a decided banded structure consisting of alternating bands of high-grade ore, milling ore, and sometimes waste. Samples are cut to correspond to the different bands; one sample is taken to each variation in the banding.

Drifts are sampled across the back at regular 5-foot intervals. If conditions warrant, such as a change of formation, samples are taken at shorter intervals.

In estimating the tonnage in a block of ore between two levels, the average width of the vein on each level is first calculated by averaging the widths of ore cut by all crosscuts on each level. The averages for the two levels are then combined into an average width for the entire block. This, multiplied by the length of the block, gives the volume in cubic feet. A ton of ore is equal to 10.5 cubic feet in place. The method used to determine the grade of ore is to calculate "average value of ore" of all openings in the vein on each level, in foot units. From these the average of the mine is readily obtained.

METHODS OF DEVELOPMENT AND MINING

Development. - The mine has been developed through five vertical shafts, numbered in sequence of sinking; also, four adits have been driven, two on the 200 and two on the 500 levels.

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Above the 2,000-foot level, which is practically at the top of the pre ent active zone, the mine is developed by levels at 100-foot vertical interval which are connected to either No. 1 or No. 2 shafts, or both. Below the 2,000 foot level there are main haulage levels at 2,250 and 2,550 feet and intermedia levels for bringing in waste filling at 2,125 and 2,400 feet. Development of main haulage level at 2,800 feet has just been started from the No. 5 shaft, a eventually there will be an intermediate level at 2,675 feet. The next lower level will be the 3,000, with an intermediate at the 2,900.

For the purpose of decreasing the cost of level development and equip- ment, the distance between the main haulage levels was increased to 250 feet and then to 300 feet. At the 2,550 level rock temperatures had increased to such an extent that driving the long stope raises required in mining the ore o the 300-foot lift was very expensive. Also, with the longer lifts it was fou difficult to keep the raises open until the stopes were finished, and, in addi tion, the chutes wore out and had to be replaced. These disadvantages, toget with the added cost of handling timber into the stopes and of ventilating the stopes before the raises were holed-though, more than offset the advantages gained. The distance between levels was therefore decreased to 250 feet, and the next level will be further decreased to 200 feet.

No. 1 shaft was started on the vein and extends to the 800 level. It is now caved above the 400 level and is used only as a return air course, al- though a considerable amount of ore was hoisted through it prior to 1915. No. 2 shaft is in the footwall at a point 400 feet north of No. 1 shaft and is connected by crosscuts to the vein on all levels except the 2,125 level. It extends below the 2,550 level.

Before a fire that occurred in 1927,[4] No. 2 shaft was used for hoistin waste from development, hoisting direct-smelting ore, and handling material an men. At the time this paper is being written, No. 2 shaft is still out of com mission, as it is being concreted. No. 3 shaft is 400 feet south of No. 1 in the hanging wall, and is connected to all levels except the 2,400 intermediate and some of the old levels above the 2,000; it also extends below the 2,550 le Before the fire No. 3 shaft was used for hoisting ore of milling grade and low ing men and some supplies; all ore at present is being brought out of the No. shaft. No. 4 shaft is about 1,000 feet east of No. 2, and is in the footwall close to the vein. It extends to the 1,500 level, but is continued as an up- cast airway by raises extending to the 2,000 level. No. 5 shaft is used to hoist development waste and handle men and material, but will eventually beco one of the main ore-hoisting shafts. It was sunk to prospect the faulted weste portion of the vein, and is in the hanging wall about 700 feet south and 500 feet west of the 2,250-level stopes. Shafts Nos. 2, 3, and 5 are used as fres air intakes.

All shaft and winze openings are rectangular in cross section and of two or more compartments. The following tabulation gives the number and size of the compartments in the different shafts:

4 Gardner, E. D., and Parker, D. J., Shaft Fires - Magma Mines. Reports of Investigations, Serial 2882, Bureau of Mines, 1928, 8 pp.

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Number and size of compartments in shafts

Shaft No.	Number of Compartments	Size of Hoist Compartments, feet	Size of manway compartments, feet.
1	2	4 x 4	4 x 4
2	3	4 x 4	(1) 4 x 3
3	3	5 x 4	5 x 4
4	3	5 x 5½	5½ x 6
5	4	5 x 4	5 x 4
21 winze	2	5 x 5	5 x 5

Manway raises are driven usually at the limits of the ore shoot for ventilation and emergency exits. Supply raises are within the separate stopes and will be discussed later in this paper.

All permanent workings are driven in the country rock. Haulage drifts, or footwall drifts, are driven in the footwall of the vein. Experience has shown that a pillar at least 25 feet thick between the footwall drift and the vein is desirable. Drifts for bringing filling into the stopes are run in the hanging wall 13 feet from the vein. Extraction drifts are driven in ore along the footwall of the vein. The country rock, and in most cases, the ore, are uniform and blocky; they are readily broken by blasting.

Drilling. - Drilling and blasting practice has been left mainly to the judgment of the boss in direct charge of that section of the mine in which the work is being done. Heavy jackhammer drills are used in shafts and winzes; a lighter type is used for breaking bowlders and plugging for timber. Drifting machines of the Leyner-type are used in all drifts and crosscuts. Some drifting machines are mounted on shells which give a 48-inch travel. With these machines a 7-foot hole can be drilled in most of the country rock with two pieces of steel, a 4-foot starter and a 7-1/2-foot finisher. Hand-rotated stopers are used in raise work and stoping. No dry drilling is done in the mine.

Three kinds of hollow drill steel are used--seven-eighths inch hexagon for jackhammers, one and one-quarter inch round for drifters, and one-inch quarter-octagon for stopers. Drifter and jackhammer steel are made up with 18-inch changes of length. Stoper drills have recently been equipped with long front-heads to decrease steel breakage and increase the life of the front-head itself. For this reason, stoper steel is made up with 12-inch length changes. A single 14-degree taper crossbit is used in all cases. The drill bits are the same for all kinds of steel; starters are 2-3/16 inches in diameter and the other bits decrease by one-eighth inch in gage with each change.

The air compressors are at the smelter, where a pressure of 100 pounds per square inch is maintained. The pressure at the drills averages about 75 pounds per square inch.

Compressed air is conveyed to the collar of the mine shafts, a distance of about 5,000 feet to No. 5 shaft and 8,000 feet to No. 2 shaft, through 10-inch

(1) Below 2,625 feet the manway also is 4 x 4.

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welded and 8-inch threaded iron pipe and undergoes a total loss of pressure of about 5 pounds per square inch. The air lines in the shafts are 8-inch 10-bolt flanged iron pipe. Four or 6-inch lines are carried through the main drifts and 2-inch lines into the crosscuts and extraction drifts. Two-inch lines are also carried into the stopes and raises.

Water under sufficient gravity pressure for efficient drilling is piped into all headings. The water mains in the shafts are of 2-inch pipe. Two-inch pipe is used for lines on the levels; branch lines into stopes and raises are of 1-inch pipe.

A repair shop for all rock drills, and air and water hose is located underground on the 2,250 level as near the center of activity as practical.

Blasting. - All development headings are blasted with 40 per cent gelatin dynamite. Occasionally hard rock is encountered where the use of 60 per cent gelatin dynamite is justified. Almost all blasting in the stopes is done with 35 per cent gelatin dynamite. Black waterproof fuse and No. 6 detonators are used for general blasting. Raises over 50 feet high and shafts and winzes are blasted with No. 6 electric delay detonators. Faces of drifts and crosscuts in which considerable water is developed and some of the long straight drift headings are blasted with electricity. The 125-volt mine circuit is used for all electric blasting. Connections to the face are made by plugging from the mine circuit through a reel carrying about 200 feet of blasting wire. A locked switch is located at a safe distance, usually at the intersection of the exit drift. For blasting shaft rounds the switch is placed near the collar. The boss, or contractor, carries the only key to the switch.

Shafts and Winzes. - At the present rate of production it is necessary to develop an additional 100 feet in depth each year. The method of procedure is to sink the three main working shafts - Nos. 2, 3, and 5 - 200 to 300 feet at a time. As soon as the shafts are down, especially No. 2 and No. 5, drifting is started on the new haulage level.

The center V-cut round is used in all sinking operations and in raises where the size will permit. From experience, definite conclusions have been reached regarding the depth of rounds that can be economically pulled in shaft sinking. In general, 6 to 8 foot rounds are allowed in sedimentary rocks and 5 to 6 foot rounds in diabase. Deep rounds in diabase tend to shatter the sides of the shaft, causing a large overbreak. The additional time required to shovel out the overbreak and the increased timbering more than offsets the advantage gained by the use of deeper rounds.

The timber used in shafts and winzes is No. 2 mine grade 10 by 10 inch Oregon fir. The best pieces from all carload shipments are set aside for use in making shaft wall plates and end plates. All shaft sets are framed at the portal of the 500 adit. Only enough sets for immediate use are kept on hand as there is danger that framed timber, especially wall plates, will twist when exposed to the weather. Figure 2 shows the details of shaft timber.

In deepening Nos. 2 and 3 shafts, timbered pilot raises were driven between several of the levels and later enlarged to the desired sizes; the broken

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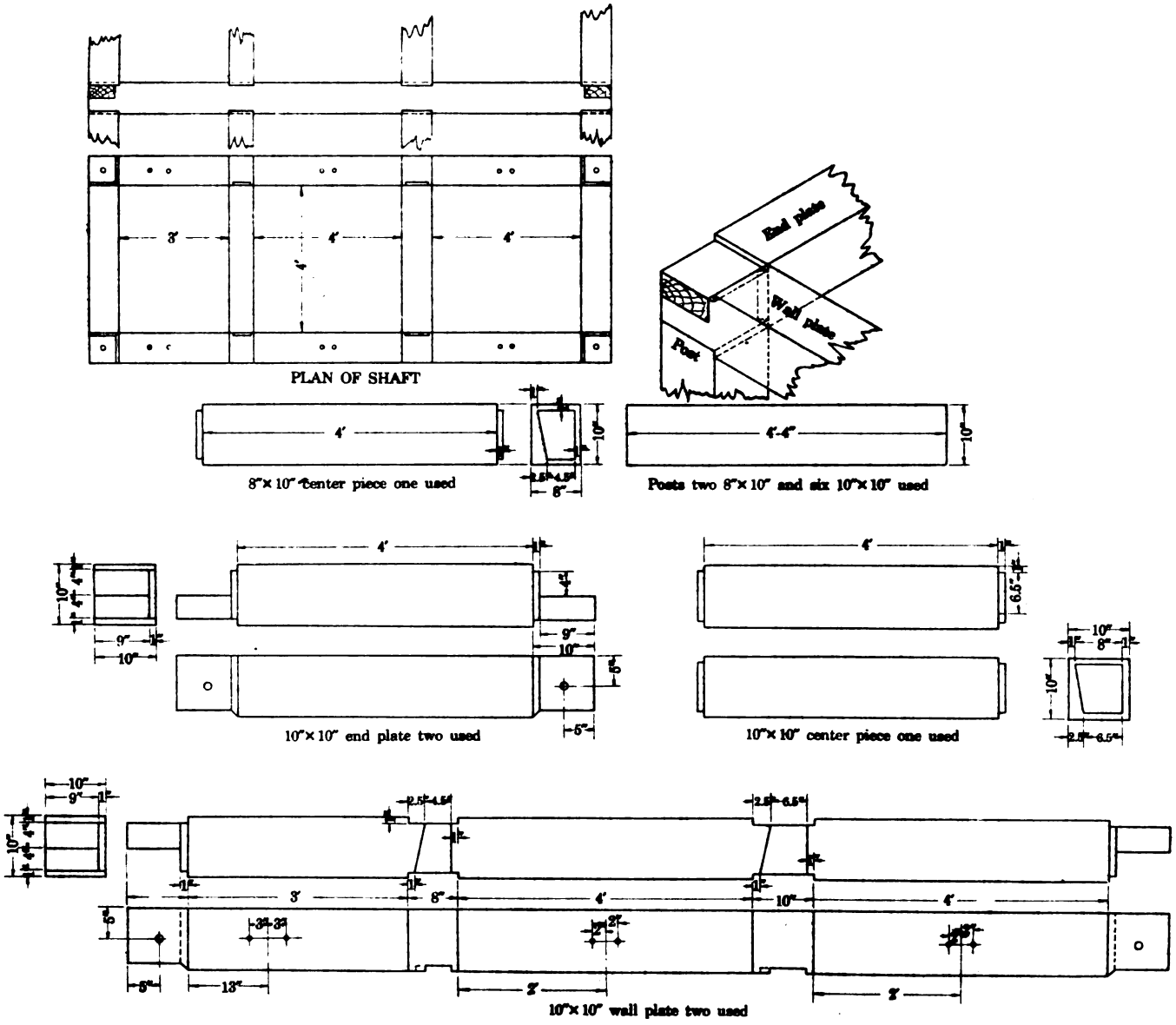
PLAN OF SHAFT
End plate
Wall plate
Post
8"× 10" center piece one used
Posts two 8"× 10" and six 10"× 10" used
10"× 10" end plate two used
10"× 10" center piece one used
10"× 10" wall plate two used

Figure 2. – Sketch of shaft set

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rock was drawn out at the bottom. This practice has been discontinued in favor of direct sinking, for the following reasons: (1) Shaft raises are necessarily a considerable distance from the source of fresh air and are hard to ventilate; (2) vertical raises 200 to 300 feet long are difficult and expensive to drive; and (3) stripping and enlarging the raises results in serious overbreak, many times requiring one to two and sometimes three stringers outside the shaft wall plates to catch up the ground. This is especially true in diabase.

Most of the sinking at the Magma mine has been done by the use of a small electric hoist, which gives good service when the hoisting depth does not exceed 800 feet below the hoist level. When a shaft is being deepened, the sinking hoist is generally installed on the lowest working level. In a three-compartment shaft the center or No. 2 and the manway compartments are closed by two bulkheads of 10 by 10 inch timber spaced 5 feet apart vertically, about 50 feet above the sinking level. The No. 1 compartment is bulkheaded on the first and the second sets below the level with 10 by 10 inch timber. A tight brattice of 2-inch Oregon fir lagging is placed between No. 1 and No. 2 compartments, extending from the lower bulkhead below the level to the upper bulkhead above. The sinking sheave is placed on 10 by 10 inch timbers, securely bolted to the shaft-wall plates. An idler sheave is hung to the brow of the shaft station in such a manner that the hoist cable will clear the shaft timber. By this arrangement No. 1 compartment is left unobstructed so that the main hoist can be operated directly to the sinking level.

Two types of sinking buckets are used. One has the bail attached to the top of the bucket. A chain about 18 inches long and having a large steel disk on the loose end is fastened to the bottom of the bucket. The second type of bucket has the bail attached to the outside just below the center. This bucket is held in a vertical position in hoisting by means of a loose steel link around the bail, which fits over a lug riveted to the top of the bucket. The buckets hold from 12 to 16 cubic feet of rock.

A trapdoor in the form of a slide or chute which is hinged to a wall plate is used with both buckets. The door is controlled by counterweights, and when lowered in place over the shaft compartment it inclines about 45°. When the first type of bucket described is used, a forked steel bracket is bolted to the center of the door near the end.

A bucketful of muck to be dumped is pulled sufficiently high to clear the door. When the door is in place, the engineer is signaled to lower. The disk which is fastened to the bottom of the bucket automatically drops into the forked bracket on the door. As the cable becomes slack the bucket is tipped forward, discharging the muck into a car or bin. The bucket is released by hoisting away from the door and is ready for the return trip to the bottom of the shaft as soon as the door is pulled up.

When using the second type of bucket the top lander releases the link and dumps the bucket by hand into a car or bin.

A drill header having water and air connections for four machines and two blow pipes and hooks to hold the same number of hose is lowered into the shaft at the beginning and hoisted at the end of the drilling shift. The hose

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are about 18 feet long and are connected to the header before being lowered. A 2-inch armored air hose 50 feet long, connected to the main air line, supplies air to the header. Steel and drills are lowered and hoisted in the bucket. Crossheads suspended 8 feet above the bucket control the travel of the bucket through the shaft. Shaftmen ride the bucket standing on the edge holding to the cable. Air and water lines are carried down as sinking progresses.

The following tabulation gives the approximate time for the different operations in shaft sinking:

	Hours
Setting up	.5
Drilling	5.5
Loading and blasting	1.0
Waiting for smoke to clear out	.5
Mucking down for timber	6.0
Timbering	3.0
Mucking and picking bottom	7.5
Total	24.0

During 1928 the advance per man-shift in sinking averaged .21 feet.

The following table shows the detailed cost figures in sinking No. 5 shaft from the surface to the 2,531-foot level:

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Cost of sinking No. 5 shaft

Footage, Month: 0
Footage, Year: 389
Footage to date: 2,531

	MONTH		YEAR		TOTAL TO DATE	
	AMOUNT	COST PER FT.	AMOUNT	COST PER FT	AMOUNT	COST PER FT
Jiggers @ $6.00 base	-	-	1,521.03	3.91	9,744.29	3.85
Shaftmen @ 5.50 "	-	-	5,893.25	15.15	41,490.67	16.39
Toplanders @ 4.13 "	-	-	2,490.39	6.40	7,503.09	2.96
Trammers @ 4.13 "	-	-	55.75	.14	1,474.62	.58
Sinking bonus - - - -	-	-	11,512.75	29.60	75,326.61	29.76
Explosives - - - - - -	-	-	1,795.22	4.62	10,420.92	4.12
Timber material - - - -	-	-	5,545.05	14.25	37,361.42	14.76
Timber framing - - - -	-	-	557.47	1.43	5,272.55	2.08
Compressed air - - - -	-	-	484.49	1.25	6,601.33	2.61
MISCELLANEOUS:						
Labor - - - - - - -	11.90	-	266.47	.68	5,040.41	2.00
Supplies - - - - -	5.26	-	316.85	.81	4,374.18	1.73
500 sinker hoist operation:						
Labor - - - - - - -	-	-	-	-	4,660.22	1.84
Supplies - - - - -	-	-	-	-	146.76	.06
Power - - - - - - -	-	-	-	-	1,199.40	.47
2,550 sinker hoist operation:						
Labor - - - - - - -	-	-	1,145.42	2.94	1,145.42	.45
Supplies - - - - - -	-	-	6.96	.02	6.96	
Power - - - - - - -	-	-	30.15	.08	30.15	.01
No. 5 hoist operation:						
Labor - - - - - - -	-	-	-	-	6,461.20	2.55
Supplies - - - - - -	-	-	-	-	806.58	.32
Power - - - - - - -	-	-	-	-	2,067.22	.82
Pumping and drainage:						
Labor - - - - - - -	-	-	47.92	.12	15,074.18	5.96
Supplies - - - - - -	-	-	2.75	.01	1,787.48	.71
Power	-	-	-	-	3,345.35	1.32
Air and water lines:						
Labor - - - - - - - -	-	-	15.88	.04	2,126.18	.84
Supplies - - - - - -	-	-	123.20	.32	4,353.88	1.72
Power lines:						
Labor - - - - - - - -	-	-	789.11	2.03	1,470.26	.58
Supplies - - - - - -	-	-	3,148.82	8.10	5,396.39	2.13
Ventilation:						
Labor - - - - - - - -	-	-	-	-	212.12	.08
Supplies - - - - - -	-	-	156.95	.40	3,481.25	1.38
Telephones and signals:						
Labor - - - - - - - -	-	-	303.05	.78	508.38	.20
Supplies - - - - - -	-	-	587.60	1.51	1,388.19	.55
Dump:						
Labor	-	-	-	-	2,872.00	1.14
Supplies	-	-	-	-	1,344.40	.53
Total	17.16		36,796.53	94.59	264,494.06	104.50

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Cost of sinking No. 5 shaft - continued

	MONTH		YEAR		TOTAL TO DAT	
	AMOUNT	COST PER FT.	AMOUNT	COST PER FT	AMOUNT	COST PER
500 sinker hoist installation	-	-	-	-	3,241.86	-
2,550 sinker hoist installation:						
Labor: - - - - - - - - -	-	-	920.81	-	920.81	-
Supplies - - - - - - - -	-	-	776.68	-	776.68	-
Pumps and motors installation:						
Pumps - - - - - - - - -	-	-	-	-	2,789.67	-
Motors - - - - - - - - -	-	-	-	-	2,137.63	-
Labor - - - - - - - - -	-	-	-	-	3,206.34	-
Supplies - - - - - - - -	-	-	-	-	3,785.44	-
Collar and sheaves:						
Labor - - - - - - - - -	-	-	-	-	849.96	-
Supplies - - - - - - - -	-	-	-	-	2,463.34	-
Towers:						
Labor - - - - - - - - -	-	-	-	-	1,201.26	-
Supplies - - - - - - - -	-	-	-	-	1,628.44	-
Change and dry room:						
Labor - - - - - - - - -	-	-	196.76	-	3,170.66	-
Supplies - - - - - - - -	-	-	-	-	303.50	-
Skips and cages:						
Labor - - - - - - - - -	-	-	285.23	-	2,935.10	-
Supplies - - - - - - - -	-	-	127.17	-	810.23	-
Total cost	17.16		39,103.18		294,714.97	116.44

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Concreting is one of the major operations in shaft work. By the end of 1928 a total of 2,900 feet of shaft had been concreted, as follows: 1,465 feet in No. 2 shaft, 350 feet in No. 3 shaft, 650 feet in No. 4 shaft, and 435 feet in No. 5 shaft. No.2 shaft will be concreted from the 200 level to below the 2,550 level by the end of 1929. No. 4 shaft is to be concreted from the 650 level to the 1,500 level during 1929. Whenever shaft timber needs to be replaced a shaft will be concreted if possible.

A concrete mixer, having a capacity of one-sixth cubic yard, driven by an electric motor, is used. The mixer is placed on a convenient level above the section of the shaft to be concreted. The concrete consists of river-run sand and gravel in the proportion of 4½ parts of sand and gravel to one of cement. Bins for the charge are built on the station high enough for the material to run by gravity to the measuring hoppers just above the mixer. Sand and gravel are lowered in end-dump mine cars by means of the main hoist and dumped directly into these bins. The mixed concrete is poured through a 4-inch iron pipe to the forms below. An advance of about 10 feet a day is made under average conditions in a three-compartment shaft. Forms are made of 2-inch lagging of standard mine length. The lagging is used only once, and when stripped is sent into the stopes for use there. All shaft timber is removed before concreting. When the overbreak is large, outside forms are used, and the space is filled with rock. The minimum thickness of concrete is 10 inches.

Figure 3 is a plan and section of shaft concrete showing reinforcing details. Figure 4 is a plan and section of the 2,550 level station, No. 5 shaft.

DRIFTS AND CROSSCUTS

A toe-cut round drilled from a crossbar is used in drift work. The use of this round permits drilling and mucking to be done on the same shift, with a resulting increase in speed of advance which is of great importance, as development work on one level is generally limited to two or three faces. In addition, the responsible contractor is enabled to supervise the mucking as well as the drilling.

The depth of round pulled is adjusted to break only as much rock as can be shoveled in time to permit placing the second drill set-up for the lower part of the round. In a 7 by 9 foot heading with hand shoveling, 6-foot rounds are usually drilled. In a 10 by 10 foot drift, employing a mechanical loader, 7-foot rounds can be used. Figure 5 shows a typical round for a 7 by 9 foot heading in medium ground. In harder ground a skimmer and extra cut and breast holes would be used, while in softer ground one of the back holes and both breast holes could be omitted.

The 500 level adits are 8 by 10 feet in the clear, run on 0.35 per cent grade in favor of the load. At present two sizes of drifts and crosscuts are run in the mine: (1) Main haulage drifts, extraction drifts, and crosscuts; (2) intermediate drifts and crosscuts.

Main Haulage. - Drifts and crosscuts on the main haulage level are broken 8 by 8 feet in untimbered sections. Where timber is necessary a battered set of 10 by 10 inch posts with a 6-foot cap placed 7 feet 5 inches above the

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tie is used. On the next level extraction drifts will be 8 feet high and 5 feet wide at the top and 7 feet wide at the bottom in the clear to allow more clearance for the mine cars. Figure 6 shows the drift sets used. Sills are not used under the sets in any class of drifts or crosscuts except where such sets are in ore. In that case sills are placed in order that sill floors may be mined later. Comparatively little timber is necessary in the country rock away from the vein. In many places the diabase tends to air-slack and slough. Gunite has proved very successful in preventing this.

Mechanical loaders operated by compressed air are used to clear out the broken rock. The principle and operation of the loader are much the same as those of an ordinary steam shovel. After the dipper is filled it is drawn back toward the shovel before being elevated. This makes it possible to clean out the round completely while the miner is still drilling from the crossbar at the face.

Short rails of 4, 8, and 15 foot lengths are laid on regular ties as the face advances. These are replaced by the permanent 30-foot rails as soon as room is made.

Until recently, sectional double track superimposed on the permanent track was used for passing cars in driving haulage drifts and crosscuts. The sections of track were 8 feet long and were made from 12-pound rails riveted to $\frac{1}{4}$-inch strap iron. These sidings were moved up as the face was advanced. Side-dump cars having a capacity of 18 cubic feet were used with this track layout. Present practice is to widen the drifts for a distance of 75 feet for double track at about 800-foot intervals. These sidings can be used later for timber-truck storage. The broken rock from drift faces is removed in 40-cubic-foot cars, hauled by storage-battery locomotives.

A crew for a main haulage drift heading consists of two miners, one of whom is the responsible contractor, two helpers, and one mechanical-loader operator. One of the helpers acts as motorman, and both assist in the operations of the loader such as moving it up and picking bottom. They also lay track and put in air and water pipes. Under ordinary conditions where the ground does not require timbering, an advance of 11 feet in two shifts is made. The average round breaks about 19 2-ton cars of rock. The cycle of operations is given below:

	Hours
Setting up	1
Drilling top of round and mucking	4
Setting up and drilling lifters	1
Loading and blasting	1
Lunch and lost time	1
Total	8

Intermediate Drifts and Crosscuts. - Intermediate drifts and crosscuts are broken 7 by 7 feet in the untimbered sections, and 7 by 9 in timbered sections. Sets for these drifts are framed of 10 by 10 inch timbers. The cap is 5 feet in the clear, and the posts are 8 feet long and battered.

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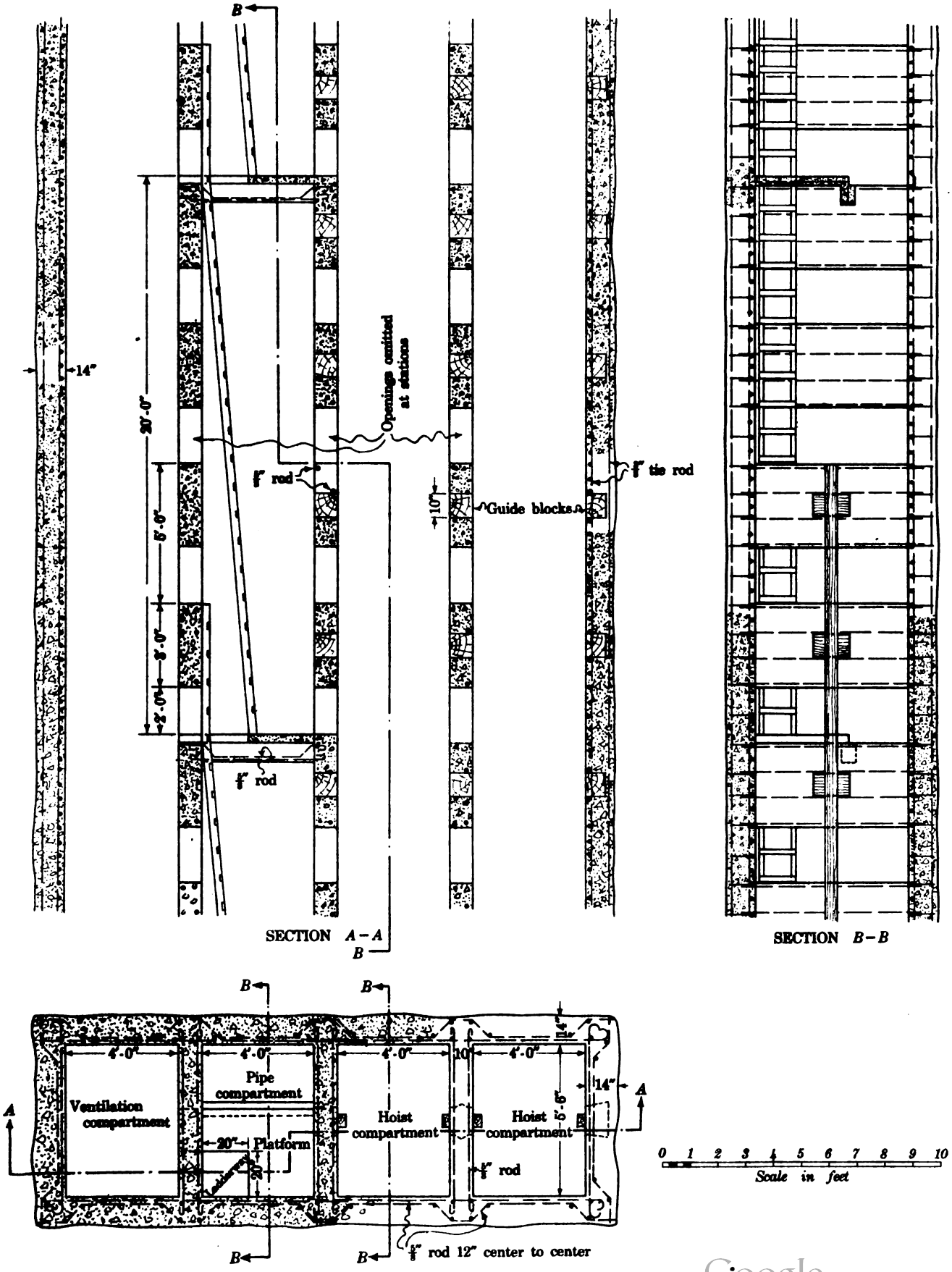

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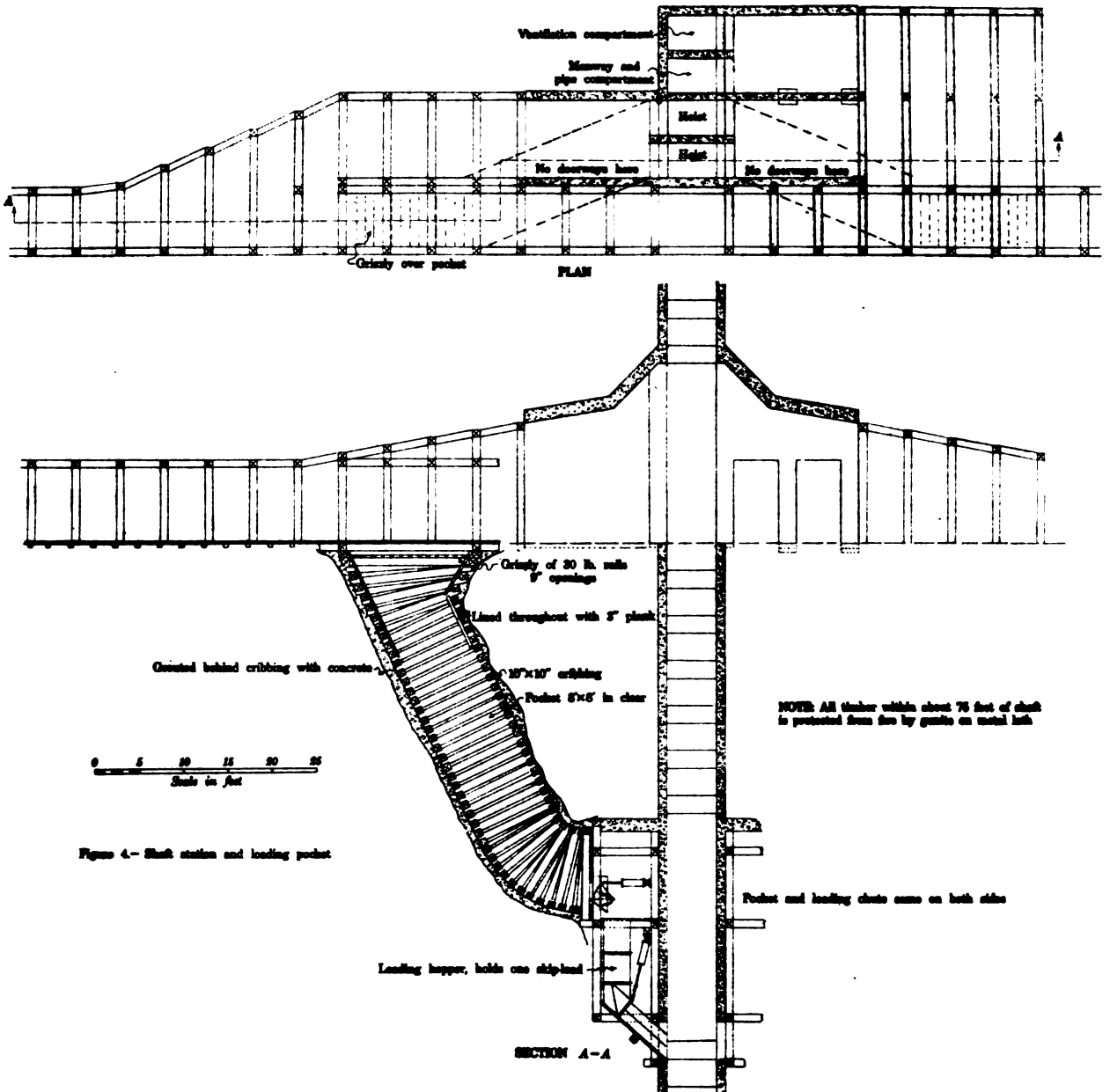
Ventilation compartment
Masonry and pipe compartment
Hoist
Hoist
No doorways here
No doorways here
A
A
Grizzly over pocket
PLAN
Grizzly of 20 lb. rails 9" openings
Lined throughout with 3" plank
Grouted behind cribbing with concrete
10"×10" cribbing
Pocket 6'×6' in clear
NOTE All timber within about 75 feet of shaft is protected from fire by gunite on metal lath
0 5 10 15 20 25
Scale in feet
Pocket and loading chute same on both sides
Loading hopper, holds one skip-load
SECTION A-A

Figure 4.— Shaft station and loading pocket

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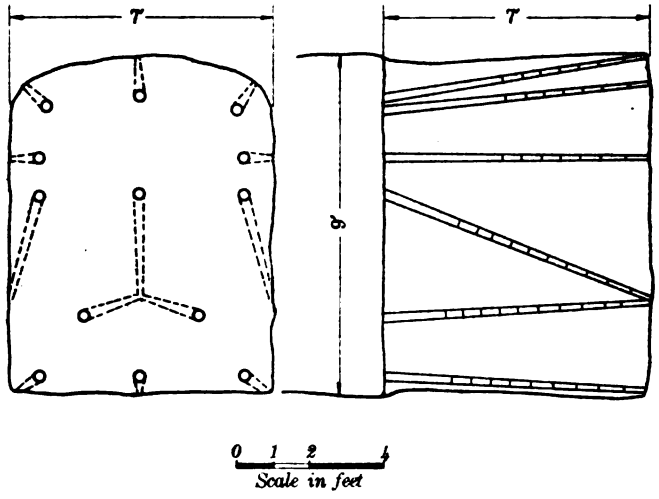

Figure 5. – Typical round for 7 by 9 foot drift

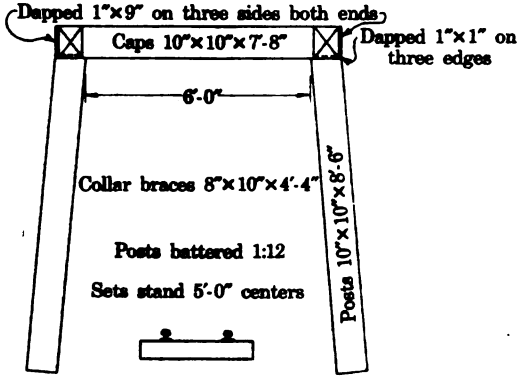

Figure 6B. – Method of timbering footwall drift (Hanging-wall drift set is the same, except that cap is only 5 ft. 0 in. in clear)

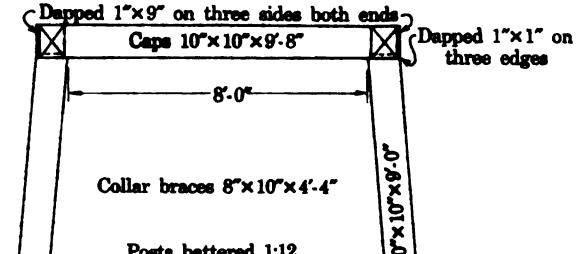

Figure 6A. – Method of timbering main ventilation and haulage drift

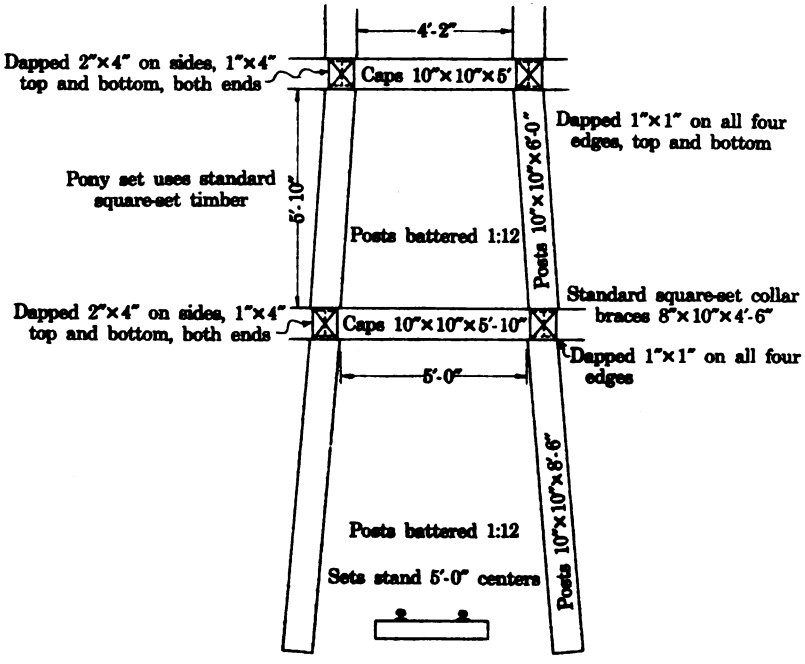
4'-2"
Dapped 2"×4" on sides, 1"×4" top and bottom, both ends
Caps 10"×10"×5'
Dapped 1"×1" on all four edges, top and bottom
Pony set uses standard square-set timber
5'-10"
Posts battered 1:12
Posts 10"×10"×6'-0"
Standard square-set collar braces 8"×10"×4'-6"
Dapped 2"×4" on sides, 1"×4" top and bottom, both ends
Caps 10"×10"×5'-10"
Dapped 1"×1" on all four edges
5'-0"
Posts battered 1:12
Sets stand 5'-0" centers
Posts 10"×10"×8'-6"

Figure 6C. – New method of timbering extraction drift. For former method see Figure 14

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The crew in an intermediate drift consists of one miner, who is the responsible contractor, and three muckers. The muckers clean out the round and lay the track and pipe. Quarter-inch steel plate is used for shoveling sheets. Usually three or four sheets 6 by 4 feet are used in a heading. Double track is laid for a short distance every 300 to 400 feet where the drift is broken wider for the purpose. Small 14-cubic-foot side and end-dump cars are used in this work. Motor haulage is provided where the tram is very long. A good crew will advance such a heading about 9 feet per day in two shifts. Twelve holes are drilled in the average round.

During 1928 the average rate of advance of all drifts and crosscuts was 0.65 feet per man-shift.

Raises. - Various types of raises have been used, such as six-post raise sets with manway and chute, stulled raises, square-set raises, and cribbed raises. Under average conditions the cribbed raise is most satisfactory and is now used exclusively. Figure 7 shows a two-compartment cribbed raise.

Most of the raises are driven in the hanging wall of the vein, which usually is crushed and contains considerable gouge. For raises other than cribbed raises it is necessary to open up a good deal of ground in order to make room for the set of timber. In hard ground the sets are frequently shot out and many times it is necessary to plug the corners to make room for the timber. Much time is lost in this way.

The advantages of the cribbed raises are that in bad ground timber can be kept close to the face, and, if necessary, with the form of timbering used, half the raise can be blasted and cribbed at a time. Less skill is required to crib raises than to put in regular raise sets.

The present practice is to drive cribbed raises of two compartments, each 4 by 4 feet in the clear. The cribbing is framed from 6 by 6-inch Oregon fir. Each set consists of eight interchangeable pieces 5 feet long over-all, dapped 1 by 6 inches on two sides at both ends. A double line of 6 by 6 inch cribbing is used between the manway and chute.

Dividers are placed in the manway to make two compartments, one for a timber slide, ventilation pipe, and air and water pipe, and the other for the ladderway. Platforms are spaced at 20-foot intervals, and ladders of 2 by 4 inch material are offset at each platform.

A raise crew consists of a timberman and a miner. Both men are considered contractors. All timber and supplies are delivered to the top of the raise by company trammers. A temporary control chute at the bottom of the raise allows the waste to be drawn into the stope. This is done by the raise man. An average round of 16 holes breaks from 3 to 4½ feet per round. The round is drilled and blasted on one shift and the timber placed on the next. Tugger hoists are used to hoist timber and supplies to the top of the raise.

For the year 1928 the average rate of advance for all raising other than that charged to stoping was 0.46 foot per man-shift.

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Preliminary Stoping Development. - The preliminary work in the development of stopes may be divided into two operations: (1) Primary exploration to determine the size and general condition of the ore body; and (2) detailed work to block out the ore body for the particular system of mining to be used.

After the station has been cut on a new level, a crosscut is driven to and through the vein. From the crosscut a footwall drift is run about 25 feet from the vein and parallel to its general course, as determined from the level above. From the footwall drift crosscuts are driven through the vein at intervals of 75 to 225 feet, depending upon the information desired. These crosscuts are later used as the beginnings of stopes. This work continues to the limits of the ore. In this manner the width and length of the ore body and the general characteristics of the ore and walls of the vein are determined, and a choice is made of the stoping method to be used.

Early Stoping Methods. - Most of the ore mined by the present company above the 800 level was stoped by the rill or "inclined cut and fill" method. Development raises were driven in the ore to the level above on 75-foot centers, the block between being considered one stope. The sill floor was then mined and a mat of double 2-inch plank laid on 8 by 8 inch stringers. The back of the stope was then mined to slope upward from the center to the raise at either end on the angle of repose of the subsequent fill. This first ore was shoveled by hand into cars on the sill floor. When the ore was all removed, two timber sets were stood in the center of the stope under the lowest part of the back to form the bottom of the extraction raise. These sets, as well as the raises at the ends of the stope, were then lagged on the outside with 2-inch lagging. Then waste from development work or the surface glory hole was run in from the level above through the two end raises. This waste was covered with a plank floor, and a 6-foot slice was mined from the back of the stope. Rounds for this and following cuts were drilled with stopers, working from the low point of the back at the center toward the waste raises at either end. When a cut was completed, the extraction raise was cribbed up to within 2 feet of the back, and the stope was again filled.

This method worked well under the following conditions: A narrow vein (5 to 15 feet wide), silicified well-drained ore, and vein-walls comparatively free from crushing and gouge. It was tried out on lower levels, but on the whole was not satisfactory, as the main ore shoot was 20 feet wide and the walls did not stand well. This method may still be used occasionally to mine ore bodies when the vein is about 5 feet wide and the walls stand relatively well. A timbered rill system was next used and proved successful; but at further depth where the vein became still wider, serious caving occurred in heavy ground. The rill and pillar system was then developed to mine the wide ore bodies.

Two grades of ore are mined from all classes of stopes; direct smelting ore carrying about 12 per cent copper, and milling ore carrying about 6.5 per cent copper. About 20 to 25 per cent of the ore mined is direct smelting ore. No attempt is made to get a close separation of the two ores. As the different bands of mill ore and smelting ore are encountered in the stopes, selective mining is used to get the desired grade. It is not necessary to clean up the stope before mining ore of a different grade, as the dilution that results is not serious.

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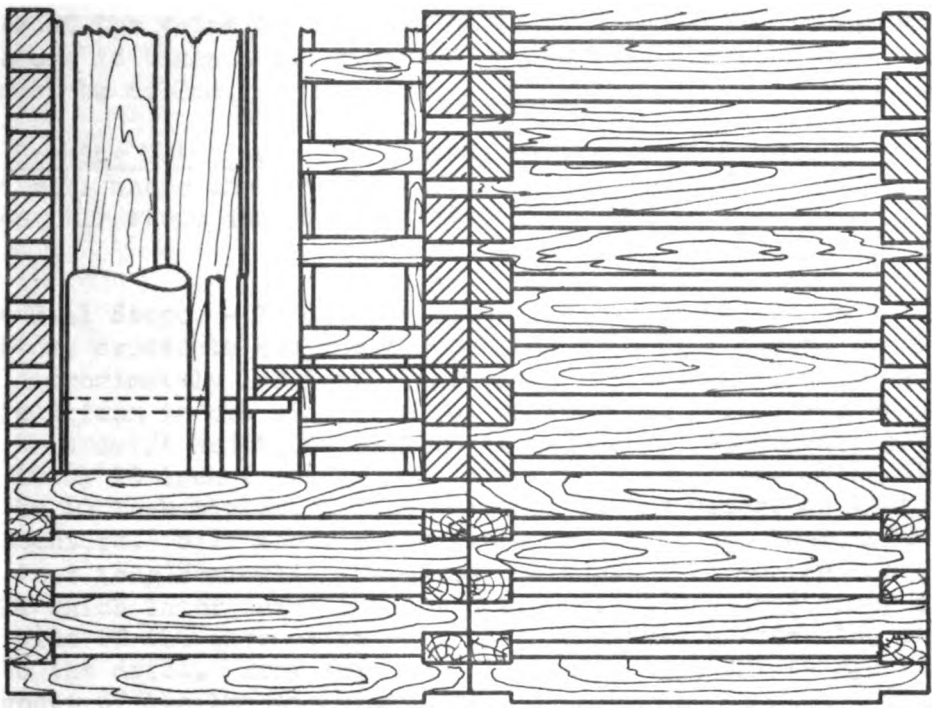
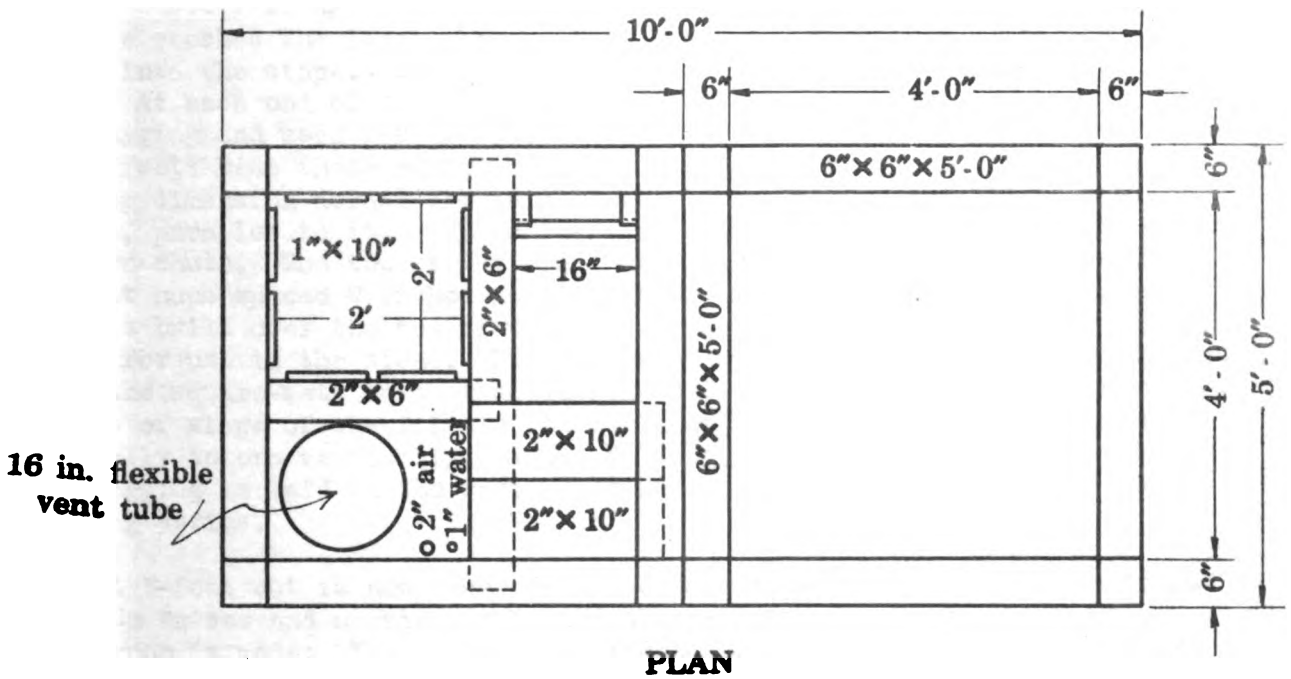


ELEVATION AND SECTION



PLAN

Figure 7. — Two compartment cribbed raise.
Not drawn to scale

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The walls of the veins at most places in the mine require immediate support when the ore is taken out. Timbered stopes soon start to take weight and all stopes must be filled.

Present Stoping Methods. - Two general mining methods are now used: (1) The timbered rill where the vein is narrow, less than 15 feet wide, and (2) a combination rill stope and pillar system where the vein is over 15 feet in width.

Timbered Rill Stope. - In developing a block of ground for the timbered rill stoping system, crosscuts are run through the vein from the footwall drift at intervals of approximately 150 feet (fig. 8). From those crosscuts an extraction drift is driven in ore along the footwall of the vein. Stopes are silled out from this drift and timbered with square-sets. Square-set posts and caps are of 10 by 10 inch timber. Caps are 5 feet long over-all; collar braces are of 8 by 10 inch timber, 4 feet 2 inches in the clear, so that posts stand on 5-foot centers. Sill floor posts are 8 feet 3 inches long, and regular posts are 6 feet long over-all. Battered posts are used in the row of sets next the footwall which later forms the extraction drift when the stope is filled. The outside of the extraction drifts is lagged to prevent the fill from running into the drift. Pony sets over the extraction drifts are also kept open on account of the loading platforms.

The length of each stope along the vein depends upon the width of the vein and the character of the ore. Generally a 30 to 40 foot stope is started and the back inclined from the top of the sill set at each end to four or five sets in height at the center. From the top floor a two-compartment cribbed raise for waste filling is driven in the hanging wall next the vein. After this raise reaches the level above, it is also used for lowering timber and supplies into the stope. The waste rock from the raise is also used for stope filling. At each end of the stope two adjoinging square-sets are lined with 2-inch lagging and used for extraction raises as the stope is mined up. Occasional offsets keep these raises next the footwall. They are oriented with their long dimension across the vein if the latter is two sets or more wide; otherwise, parallel to it. One compartment is used for a manway and the other for an ore chute. The top of the ore chute is protected by grizzlies made of square-set caps spaced 9 inches apart. After each filling of the stope a new grizzly is built over the top of the chute, the timber from the old one being salvaged for use in the stope. Filling is stopped before the waste covers the tops of the square-sets so that succeeding sets can be placed without digging. The angle of slope of the fill is the angle of repose, or, roughly, two sets horizontally to one vertically. After filling, a floor of 2 by 10 inch by 5-foot lagging is laid on the waste, and pieces of lagging are used for sills or nailing strips.

A 7-foot cut is now taken from the back starting at the ends over the extraction raises and working up toward the center. Square-sets are stood as soon as room is made. The broken ore slides down the floor and into the extraction chutes. Little shoveling is necessary. Some breaking with hammers is required to pass the large pieces through the grizzlies. Usually three such cuts are taken before the stope is again filled.

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Succeeding stopes are generally 15 to 25 feet long. The stopes are silled out as before, and the back is given a downward slope away from the filled stope corresponding to the angle of repose of the waste. This opening, as before, is timbered with square-sets. Waste is then run into the stope through the extraction raise of the previous stope, after which a floor is laid. The stope is mined to the level above by alternately taking two or three 7-foot cuts of ore and then filling. A raise consisting of two square-sets is taken up on the far end of the stope for an ore chute and a manway.

A series of stopes may be worked at one time. Occasionally waste filling may be run from one stope into the next.

A variation of this method is used where the ground will stand sufficiently well. (See Fig. 9.) The vein is silled out as before, but for a length of 10 sets and a height of two sets above the sill. This space is timbered, and the back is pyramided up to a height of about eight sets in the center, from which point a waste raise is run to the level above. Just enough ore is mined by this first operation so that the waste from running the raise fills the space. Mining now proceeds as before until the ends of the stope are about eight sets high. Then "wing" stopes are started at each end, four sets long, and mined up eight sets. Filling for the wings is run in over the filling of the center stope. In this way a length of 18 sets or 90 feet along the vein is mined and filled with only one waste raise. When the center of the stope reaches the level above, and the filling will no longer run by gravity to the ends of the stope, short raises are put up to the level above near the ends of the stope. When using this method, pillars of ore 15 feet long are left between adjoining 90-foot stopes to prevent possible subsidence of the hanging wall and injury to the hanging-wall drift. The pillar is afterward recovered by the slicing method described subsequently.

Rill Stope and Pillar System. - This system consists of a combination of 16-foot fill stopes alternating with 14-foot pillar stopes. (Fig. 10). An extraction drift is driven as before in the ore along the footwall of the vein. From this drift crosscuts are driven to the hanging wall across the vein, on 30-foot centers. These crosscuts are then widened out 8 feet on each side from the center line of the crosscut, and blasted down to a height of 10 or 12 feet, forming the sill floors of the rill stopes. The length of each stope is the width of the vein at that point, which may vary from 15 to 60 feet. The reason for adopting a width of 16 feet in the rills was that experience showed this width to be the maximum that could be safely carried in an open stope.

Stringers of 10 by 10 inch timber 16 feet long are placed on the sill floor parallel to the vein and at 5-foot centers across the vein; a double 2-inch plank floor is laid. The extraction drift is timbered with battered sets. From this drift the three sets next the footwall of each 16-foot rill stope are carried up as a three-compartment raise as mining progresses. The outside sets are for ore chutes and the center set for a manway. These ore chutes and the manway are used later for extracting the pillars.

Square-sets are placed across the vein on one side of the rill stope on the sill floor, and single 8-foot posts of 10 by 10 inch timber are stood across the vein on the opposite side to form gob lines. When the stope is

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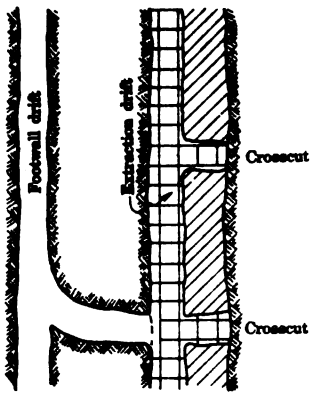
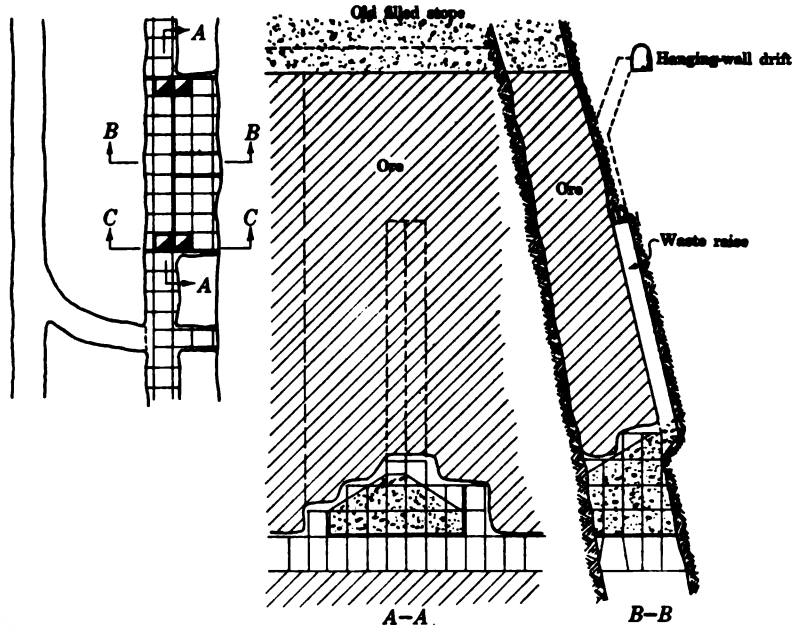
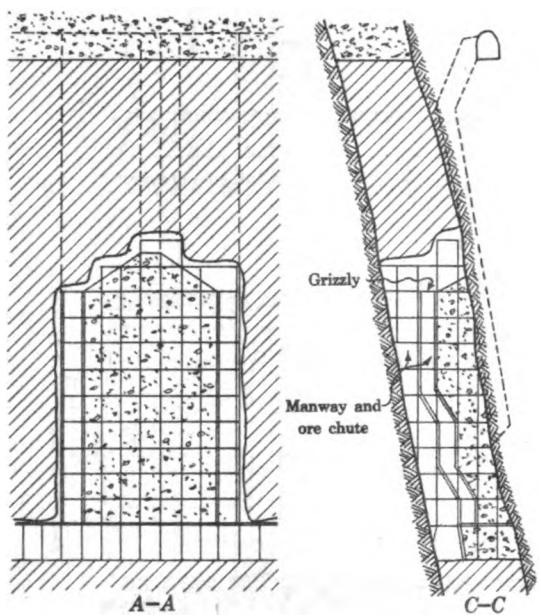


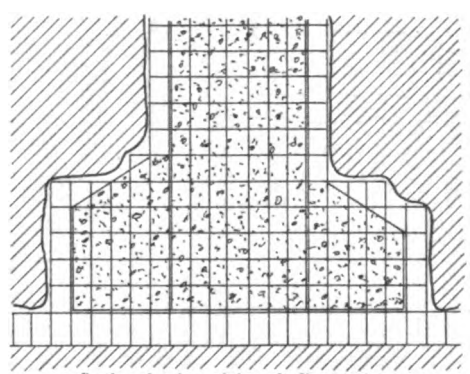
Plan, footwall drift, extraction drift, and crosscuts



Plan and sections showing chutes installed, waste raise started, first filling completed



Waste raise holed to level above, stoping progressing



Section showing mining of adjacent stopes

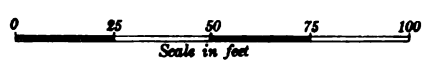


Figure 8. – Timbered rill method

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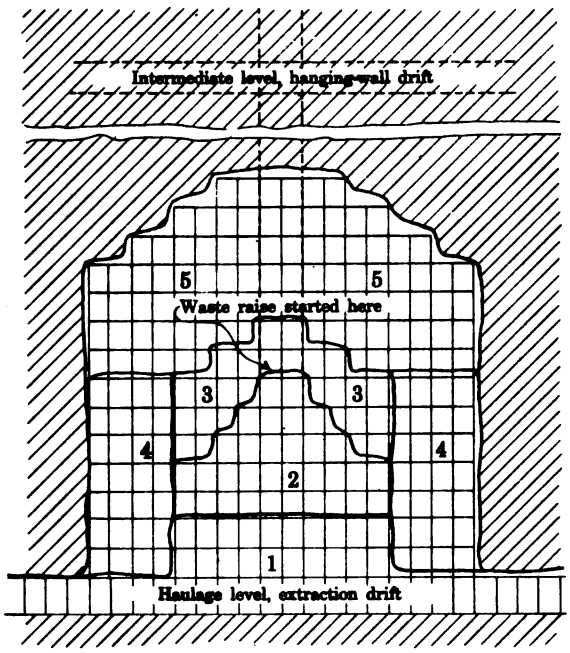

Figure 9. – Modified timbered rill method.
Numbers indicate successive stages of stoping

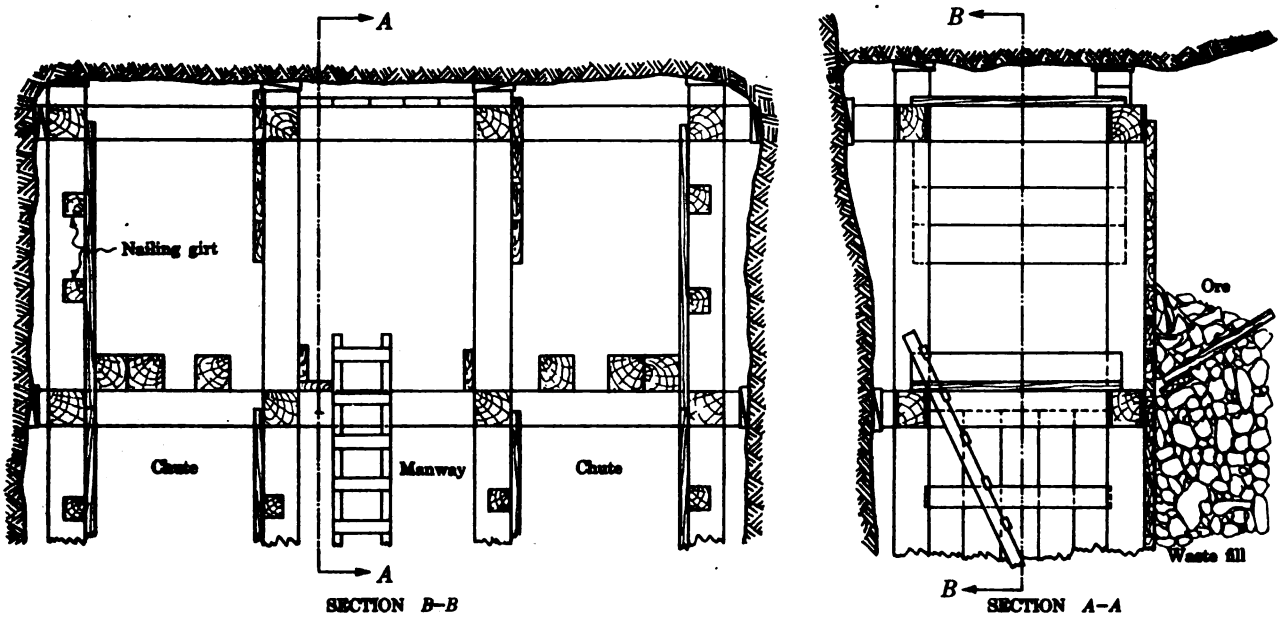

Figure 11. – Manway, extraction chutes, and grizzlies

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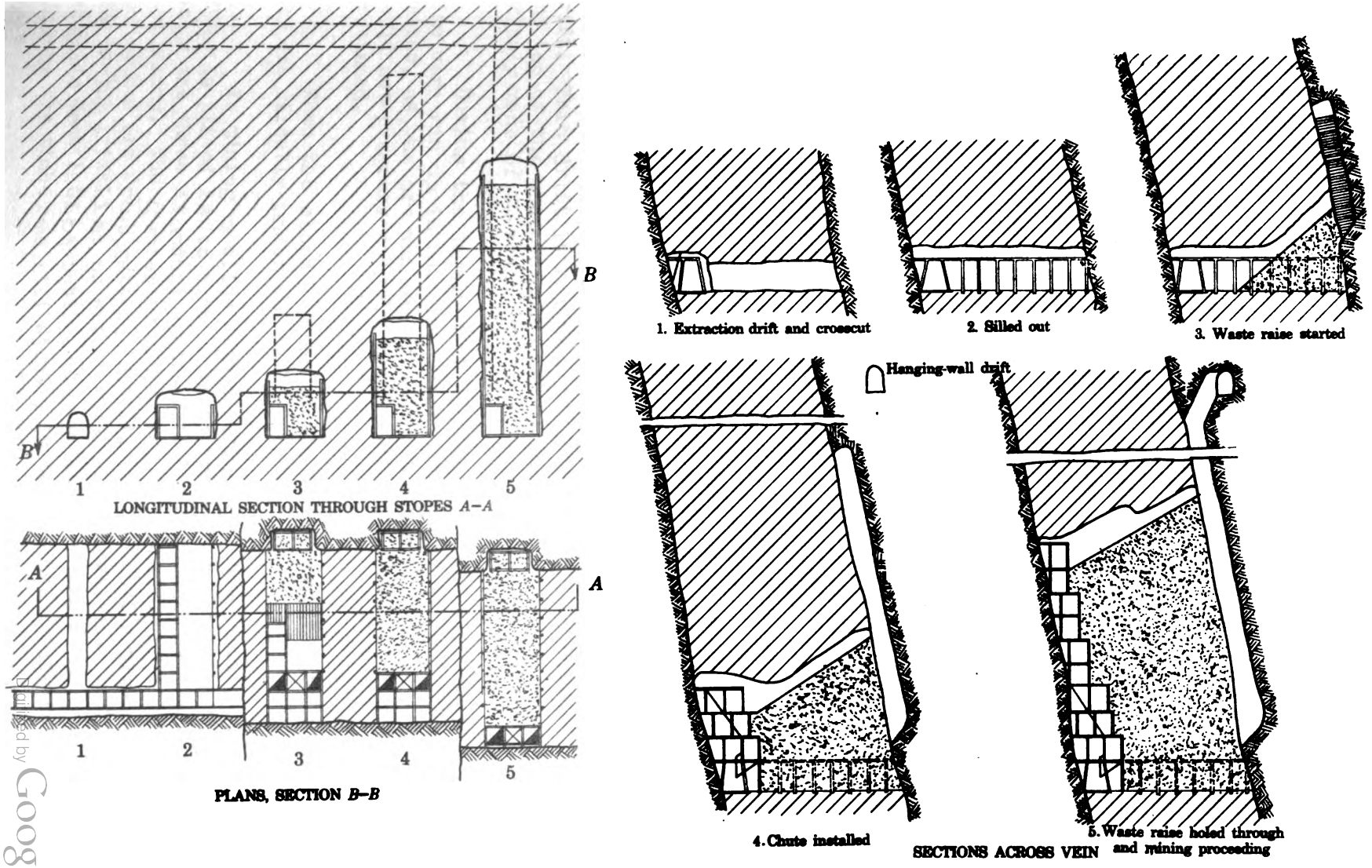

Figure 10. – Rill stoping in rill and pillar system

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filled the square-setts are held open as a crosscut, which affords a means of access to future stoping operations from below and also to the hanging-wall drift. A bridge post of 10 by 10 inch timber is placed outside all sill posts which support waste fill, and the gob lagging is nailed to this bridge post. The space between the two posts is 4 inches.

After all sill floor timbering is in place, a cribbed raise is started in the hanging wall of the vein about 12 feet from the bottom of the sill. The broken waste from the raise is dropped into the stope for filling. No attempt is made to spread this fill. When its natural slope brings it to about 4 feet from the back, a floor is laid on the waste, and a 7-foot cut of ore is mined. The back of the stope is inclined upward to the hanging wall, its slope roughly paralleling the angle of rest of the waste fill, which is about 35°. Mechanical shovels are used to load the ore into cars on the sill floor, until the stope is high enough so that the ore can be run by gravity into the extraction raise. Dragline scrapers have also been used for this purpose.

After the broken ore is cleaned out, new gob lines consisting of single posts end on end and 2-inch lagging are placed along each side of the rill stope. The ends of the posts are framed in half, and a single strand of old hoist cable is used to tie together corresponding posts on opposite sides of the stope, the cable passing around the joints of the posts. The floor is taken up and used in the gob line.

The waste raise advances only as fill is needed in the stope, usually reaching the level above when the stope is about 35 feet high on the hanging-wall side. After this, waste from other sources is dropped into the stope as required. The waste raise is also used for lowering timber and supplies into the stope. Only in exceptional cases, as, for example, when it is required for ventilation, is an attempt made to keep the raise open below the level of the stope fill. When the waste raise approaches the level above, it is inclined away from the vein so as to connect with the near side of the hanging-wall drift. This drift is run to leave a 13-foot pillar between it and the vein. The top of the chute compartment of the waste raise is covered with a 9-inch grizzly made of 30-pound rails.

During the development of the mining method, raises for filling at one time were run up in the ore, but on account of the extremely high cost of such raises, as the ore was harder than the diabase, and for the reason that stoping could not progress very far until filling was available, the practice was discontinued.

When the stope approaches the level, square-setting is used to extract the ore directly under old filled stopes.

In mining a slice from a rill stope, the first operation after the floor is placed is to blast a round from the back on the footwall side and stand the three standard square-sets which constitute the extraction raises and manway. The center or manway set is lagged up on the stope side to prevent the broken ore from getting into it, and to provide protection for the men working at the grizzlies on either side. The manway also has room to carry 16-inch flexible ventilation tubing. The grizzly, of the same construction as that used in the timbered rill stopes, is then raised up one set to the level of the

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floor. (Fig. 11). The grizzly bars are standard square-set caps, lightly blocked in place, leaving approximately 9-inch openings.

On account of the difficulty and cost of driving the long hanging-wall raises, especially when they have to be stopped and started according to the need for filling the stope, a modified form of the above method is being used on the 2,550 level, where the ore is harder and the walls stand better. The stope is silled out and square-setted to a height of about 50 feet at the hang wall side. (See Fig. 12.) From this point the waste raise is driven to the level above. In this way it can be pushed through without interruption, and will furnish just enough waste to fill the ground already stoped. From this point stoping proceeds as before. In this method, mechanical loaders are used to load the ore from the sill floor, the work of silling out being done by heading crews who have two faces each. The sill is driven full-sized, 8 by 10 feet. When the sill floor timber has been placed, a switch is run into the stope, a floor is laid above the sill, and temporary chutes installed to draw the ore from the square-setting operations into cars on the level. After completion of the square-setting and upon filling the stope, these chutes are abandoned and the sill floor is also filled, with the exception of the cross-cut mentioned before.

Waste raises above an intermediate level start from the hanging-wall drift and are generally connected to the main level above before stoping start above the intermediate. The rock from these raises may be used in filling the stope below, or may be drawn into cars on the intermediate and trammed to other stopes.

The stopes are carried up the full lift between the two main haulage levels. The intermediate levels are used for bringing in filling and supplies and as a means of access to the stopes.

Drilling is done with hand-rotated stopers. Holes are spaced about 3 feet apart. Care is taken to drill the holes in such a manner as to get the minimum amount of shattering of the back and to establish an arch from the pillars to the center of the stope. Comparatively little block-holing is necessary in the stopes. Boulders are broken with hammers to a size to pass through the grizzlies.

A crew for a rill stope consists of two miners who do all the work, including drilling, timbering, and shoveling. Very little shoveling is necessary, because most of the broken ore runs to the grizzly by gravity.

The best working conditions are obtained when enough broken ore is left on the floor of the stope to provide secure footing for the drilling and to enable the miners to reach the back at all times to bar down loose rock.

The pillars between the filled rill stopes are extracted by a slicing method which is a modified form of the Mitchell slicing system used in Bisbee. In using this method the top of the pillar is shot out, and a line of segment sets is placed immediately below the filled sill floor of the finished stope above. (See Figs. 13 and 14.) If it seems desirable two floors under the top floor may likewise be timbered with segment sets. In this case an inverted segment set is used for the second.

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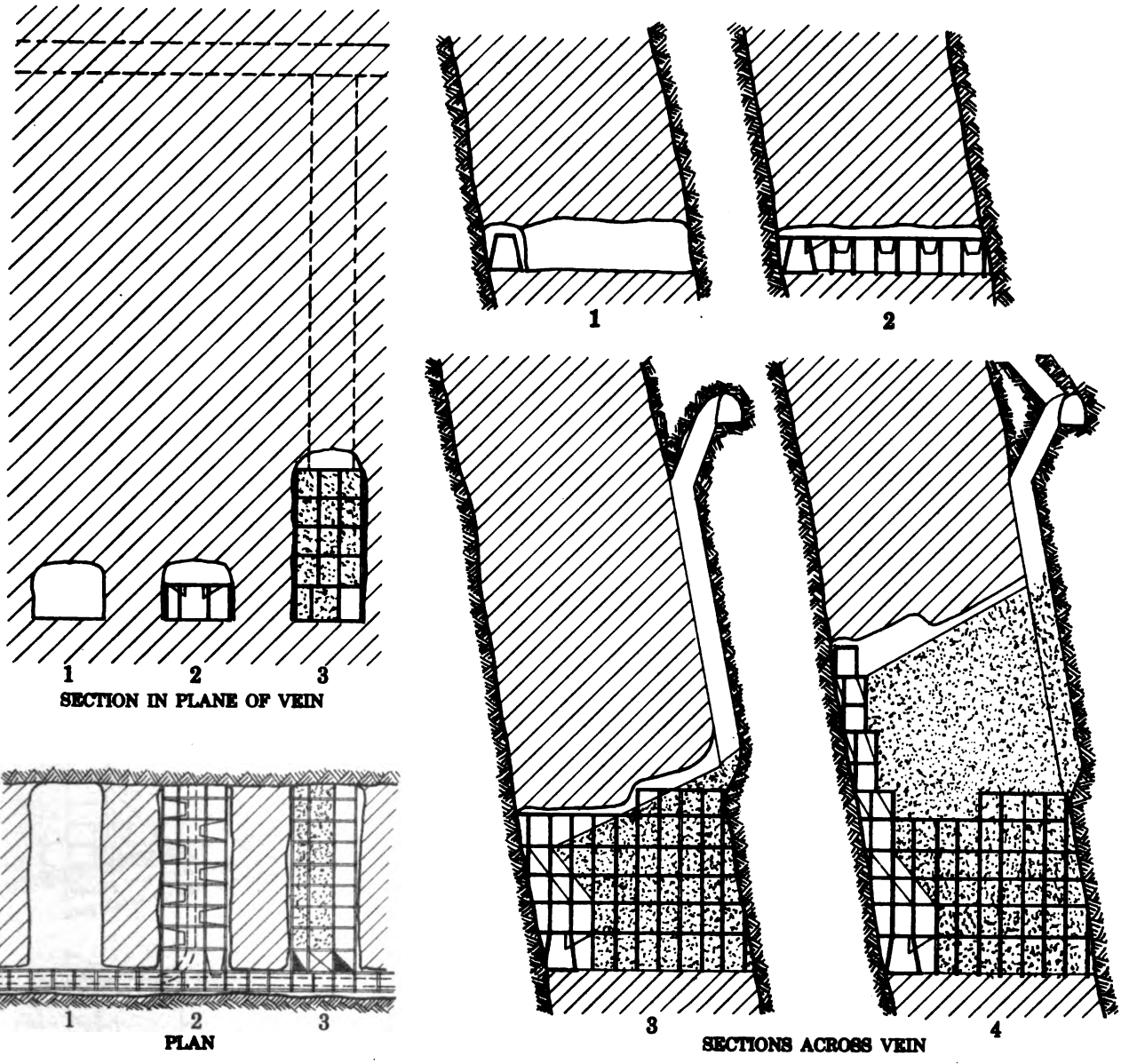

Figure 12. – Modified rill stope for stope and pillar method

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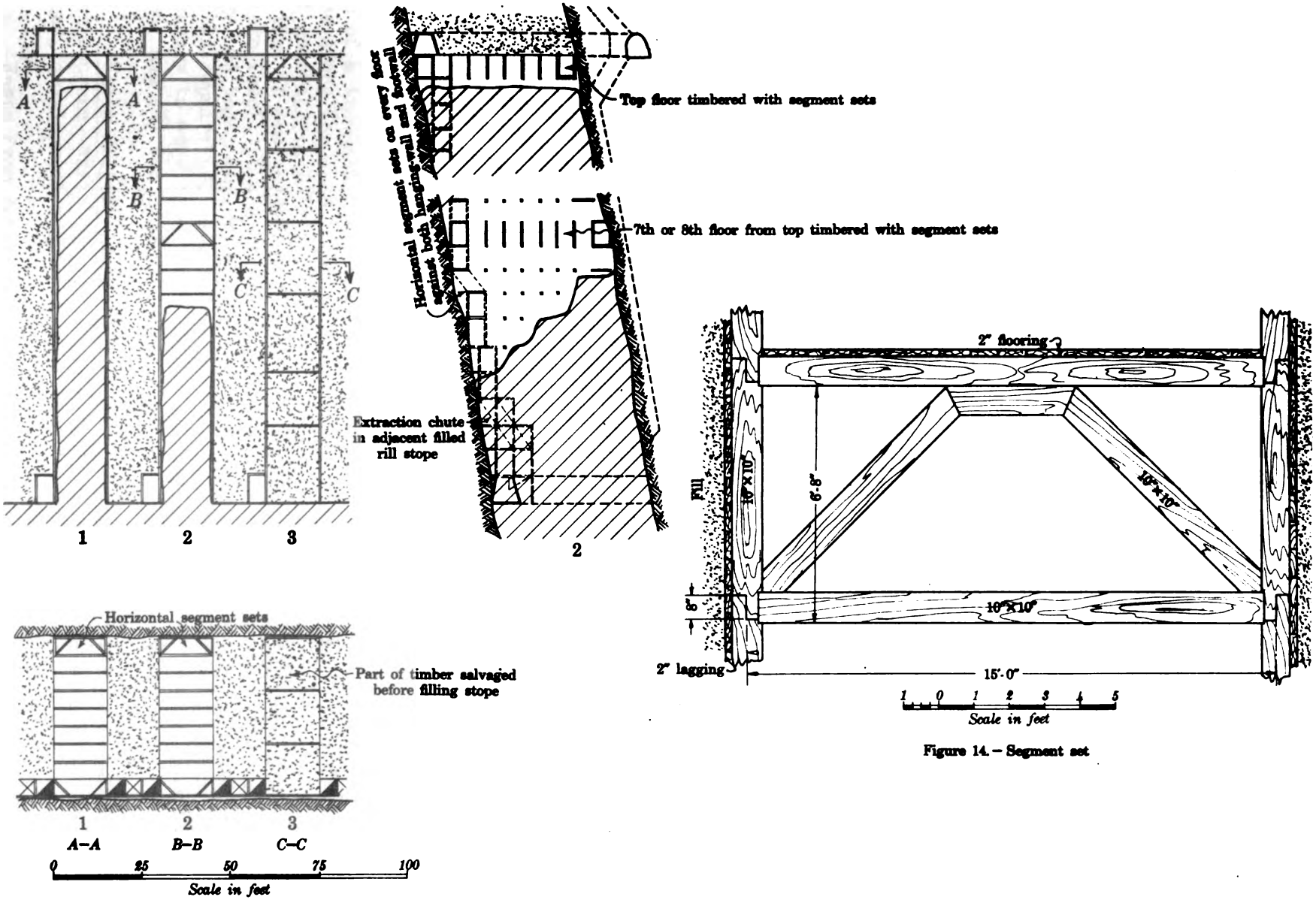

Figure 14. – Segment set

Figure 13. – Method of mining pillars in rill stope and pillar system

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The pillar is mined on about a 40° slope down toward the footwall; the broken ore runs by gravity into the square-set raise which was used in the adjoining rill stope. As stoping progresses 10 by 10 inch stringers are placed across the stope every 7 feet vertically, and upon 5-foot centers horizontally between corresponding rill stope posts. Only enough ground is broken at one time to make room for one stringer.

Drilling is done with jackhammers and practically all holes are drilled flat from staging nailed on the stringers.

Grizzlies of the same type as those used in the rill stopes are lowered to the set below each time a new cut is started. A ladderway is carried down on the stringers as mining proceeds. Both manways in the extraction raises are also maintained. Careful inspection of all stringers in the pillar stope is made during each shift as mining proceeds downward. A light floor is carried across the stope a few sets above the miners to guard against the falling of rock that may have been left lying on some of the stringers.

In addition to the segment sets placed at the top of the pillar, horizontal segment sets are placed against both footwall and hanging wall every 7 feet vertically and are lagged tightly. These sets are similar in construction to the vertical sets. As a precautionary measure, floor vertical segment sets are also placed 50 feet below the top; then if the ground starts to move, a floor can be laid at this point and the top of the stope filled with waste. A sill floor is laid at the intermediate level and the stope is filled. The lower half of the pillar is then mined in the same manner. About 75 per cent of the stringers are salvaged as the filling is put in. The crew for a pillar stope consists of two miners and one mucker. A good crew will mine from 16 to 25 tons per man per shift in an average pillar.

Branch Ore-Gathering Raises. - To extract the ore between the 1,800 and 1,600 foot levels, a series of large incline raises 8 feet by 8 feet in cross section were driven at an inclination of approximately 52° from the 2,000-foot level footwall or haulage drift to cut the footwall of the vein on the 1,800 level. These raises are shown in Figure 1, the longitudinal projection of the mine. From these main incline raises, branch raises of the same inclination were driven to cut the footwall at the center of each rill stope, or in the case of a timbered rill stope, at each end. This arrangement gave a gathering raise along the vein every 30 to 45 feet. The advantages of this system were that generally a full train or more of ore could be loaded from a single chute, and that it was necessary to maintain only two or three loading chutes on the haulage level. The system has been discontinued because (1) rock raises were large and began sloughing; (2) large timbered raises were costly and hard to keep in repair; (3) in order to get a good gathering system of raises it was necessary to develop a haulage level at least 200 feet below the level to be mined; and (4) the handling of two classes of ore was difficult.

Waste-Filling Operation. - Waste for filling stopes is obtained from three sources--waste raises in the hanging wall of the vein, previously mentioned, development work, and a waste glory hole.

Waste from development is dumped directly into the individual stope raises when convenient.

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A main waste raise extends from the surface glory hole to the 2,250 level, with drawing-off chutes on all the levels below the 1,300-foot level. At present most of the waste filling is being done from the 1,900 and 2,000 foot levels.

Development waste hoisted in No. 2 shaft is directed through branch raises into the main waste raise. Development waste hoisted in No. 5 shaft is handled in small cars and is taken off on the levels where it is needed fo stope filling.

About 50 per cent of the waste used for filling comes from the waste glory hole on the surface. An 8 by 8 foot raise was driven to the surface at point north of the vein and east of No. 2 shaft. At a point about 50 feet above the 200 level, a bulldozing chamber was mined out and a grizzly of 60-pound rails installed. The rails are crossed to form a grid with 16-inch square openings. At the 200 level a Blake-type 8 by 30 inch jaw crusher was set up, driven by a 75 hp. motor. Two men drill and blast in the waste pit and also do the bulldozing and crushing. One miner and a helper are occupied most of the time keeping the waste moving in the waste raise. When the raise hangs up, small explosive charges on long blasting sticks are used to loosen it. The waste ordinarily runs freely, but may pack in the raise if allowed to stand, especially if it is wet. Rain water from the glory hole is diverte from the main raise just below the pit through a branch raise to the 200 leve

Lime Quarry. - About 150 tons a week of limestone for the smelter flu is furnished by another surface glory hole. This is similar in operation to the waste quarry, except that no crusher is used; the rock is passed through a smaller grizzly, drawn off on the 500 level, and trammed by storage-battery locomotive to bins near the portal of the adit.

Comparative Direct Costs of Stoping. - The following tabulation gives a comparison of the direct costs of the different types of stopes. Several average stopes of the different types were used to obtain the costs given:

Comparison of direct costs of different types of stopes

	Labor				Supplies				
System	Min-ing	Muck-ing	Timber	Bonus	Waste fill	Explo-sives	Timber and material	Tons	Total cos per ton
Timbered rill	0.46	0.05	0.53	0.08	0.13	0.06	0.48	5,246	$1.79
Rills	.32	.014	.40	.17	.03	.08	.34	18,556	1.35
Pillars	.26	.053	.38	.22	.23	.04	.28	43,035	1.46

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UNDERGROUND TRANSPORTATION

Hoisting. - The three hoisting shafts, Nos. 2, 3, and 5, are equipped with double-drum electric hoists. The following table gives data about the three hoists:

Comparison of Nos. 2, 3, and 5 hoists

Hoist	Dia. of cable, inches	Dia. of drum, feet	Winding face	Rope speed, Ft. per min.	Size of motor, hp.	Skip capacity, tons	Depth limit, feet
No. 2	1	5	Smooth 36-inch	800	250	2	2,500
No. 3	1¼	7	Grooved 72-inch	1,000	400	3	3,500
No. 5	1¼	7	Grooved 108-inch	1,000	500	3	4,000

The hoist at No. 2 shaft, having reached its depth limit, is now being replaced by a new one. The new hoist will be placed on the 200 level, on the long center line of the shaft. A grip sheave of the type used for aerial tramways will take the place of a drum. The rope from a single ore skip of 5 or 6 tons capacity will pass over a sheave a few feet below the level and up a short incline to the bottom of the hoist sheave. It will make a half turn around this and pass over another sheave and down to the counterweight. The ore skip will operate in the end or No. 1 compartment, and the counterweight in the center or No. 2 compartment.

All ore skips carry a man cage fixed permanently below the skip. In addition at No. 3 shaft, through which most of the men enter the mine, units consisting of two-decked man cages are slung below the regular skip and cage for lowering and hoisting the shift. During the ore hoisting period, these units are removed at the 200 station. The guide shoes, located near the bottom floor, are hinged in such a way that the cages can be released from the guides and pulled out onto tracks on the station, for which purpose they are equipped with small wheels under the bottom deck.

For each shaft two shaft pockets having a capacity of approximately 125 tons each, are cut on all haulage levels, and are equipped with loading hoppers holding exactly one skip load. (See Figure 4.) The hoppers are loaded through over-cut arc type air operated doors. Steel grizzlies, spaced 10 inches apart, are installed over all shaft pockets. Two large inclined raises or storage pockets of about 1,300 tons capacity at both No. 2 and No. 3 shafts connect the 500 tunnel level with the skip dumps on the 200 level. These pockets originally were about 8 by 8 feet in section, and had capacities of 1,000 tons each. However, wear has increased their capacity to about 1,300 tons. Ore hoisted through No. 5 shaft is dumped into a large ore pocket just above the 500 tunnel level. The capacity of the pocket is about 300 tons. (Fig.15.)

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Haulage. - All tracks are of 18-inch gage and are laid with 30-pound rails on 6 by 6 inch redwood ties 3 feet long. The grade is one-half of 1 per cent for drainage toward No. 3 shaft where the main pump of the mine is installed. Turnouts are driven on a 50-foot radius.

Trolley and storage-battery locomotives are used for haulage through the mine.

Direct current of 135 volts is used for all trolley locomotives, and supplied by small motor generator sets. Three of these sets have been instal one each at the collars of Nos. 3 and 5 shafts, and one on the 2,550 level at No. 3 shaft.

500-Level Haulage. - From the No. 3 shaft raises the concentrating or is hauled by trolley locomotive to the mill, a distance of about 4,000 feet, 1,700 feet of which is the distance from the No. 3 shaft to the portal of the 500 adit. The trolley locomotive used for this service weighs 4 tons, has a draw-bar pull of about 2,000 pounds, and a rated speed of about 5 miles per hour. It is driven by two 20-hp. motors.

Gable-bottom cars of 3.6-ton capacity are used to haul the ore to the mill; these are standard mine cars with the sides built up 12 inches. One crew consisting of a motorman and two brakemen haul from 800 to 1,000 tons in 8 hours from No. 3 shaft to the mill.

On the 500 level 4½-ton storage battery locomotives are used to haul direct smelting ore, limestone for the smelter, timber, and mine supplies. This locomotive has two batteries so that one may be charging while the other is in use; its rated speed is 7 miles per hour. Three crews of two men each a required for this work. The direct smelting ore is trammed to a receiving bi on the railroad near the portal of the 500 adit. From there it is taken in railroad cars to the smelter.

The main shaft inclined raises are equipped with double chutes at the 500 level. One chute has an air-operated gate, similar to those used in the skip loading pockets. The other chute has a hand-operated gate of the Verde type (Fig. 16) which can be used if the air supply fails.

Timber and supplies sent into the mine are transferred from surface timber trucks to the cage or skip at No. 2 or No. 3 shafts. The cage is then lowered and unloaded on the different stations. Special crews of two men each deliver the timber by hand-tramming to the different working places. At present, however, most of the timber for use below the 2,000 level is lowered through No. 5 shaft. Recently this shaft was equipped with the timber cage and truck devised by J. W. Scott, General Foreman of the Copper Queen mine, Bisbee, which is proving very successful. This device consists of a combination cage and truck that runs on the mine tracks and can be lowered in the shaft. After the timber is loaded at the surface, the truck is taken to the shaft and lowered, after which it is trammed to the place where the timber is to be used. Details of this method of taking timber in the mine are to be given in the paper on "Mining Methods at the Copper Queen Mine."

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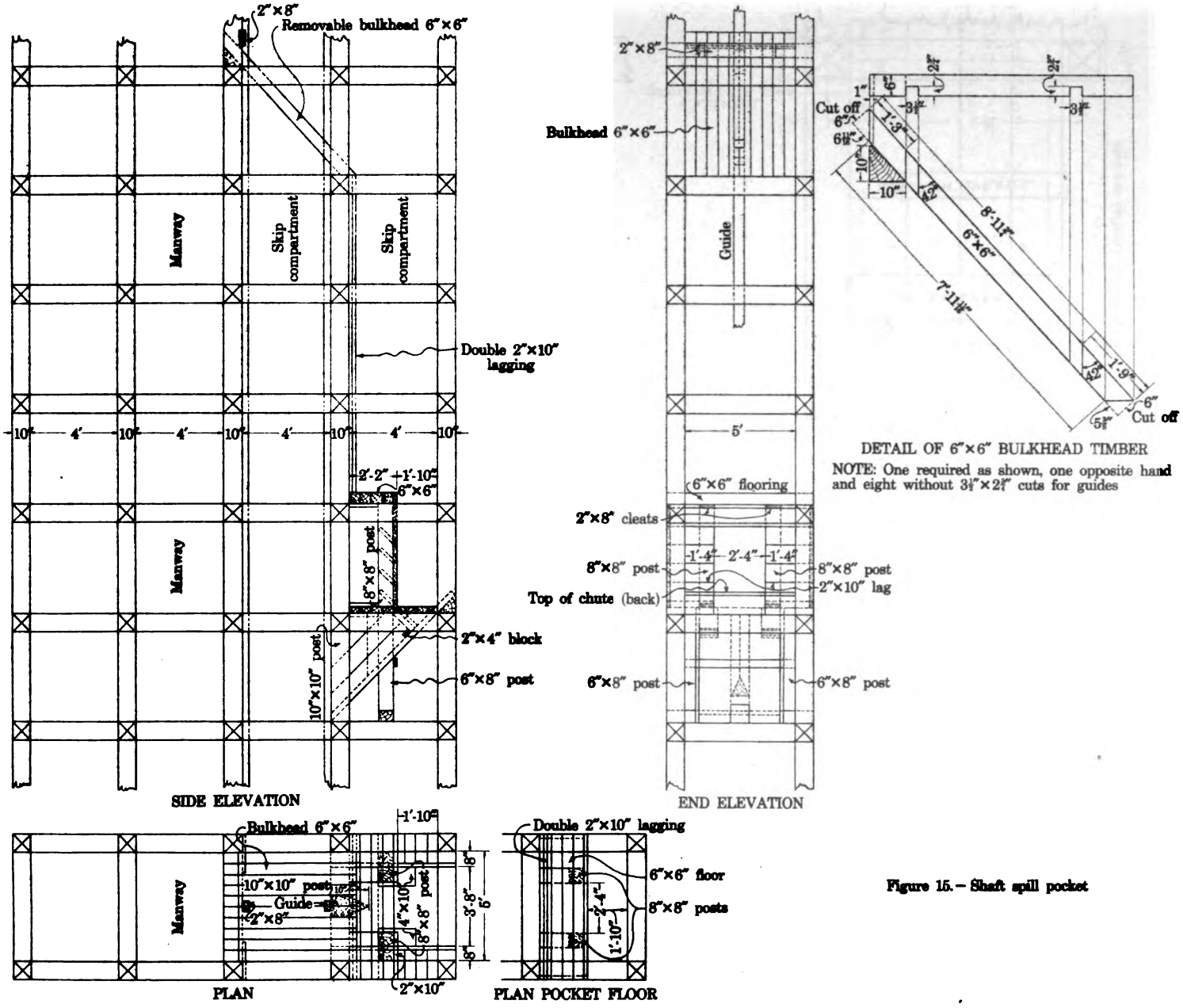

Figure 15. – Shaft spill pocket

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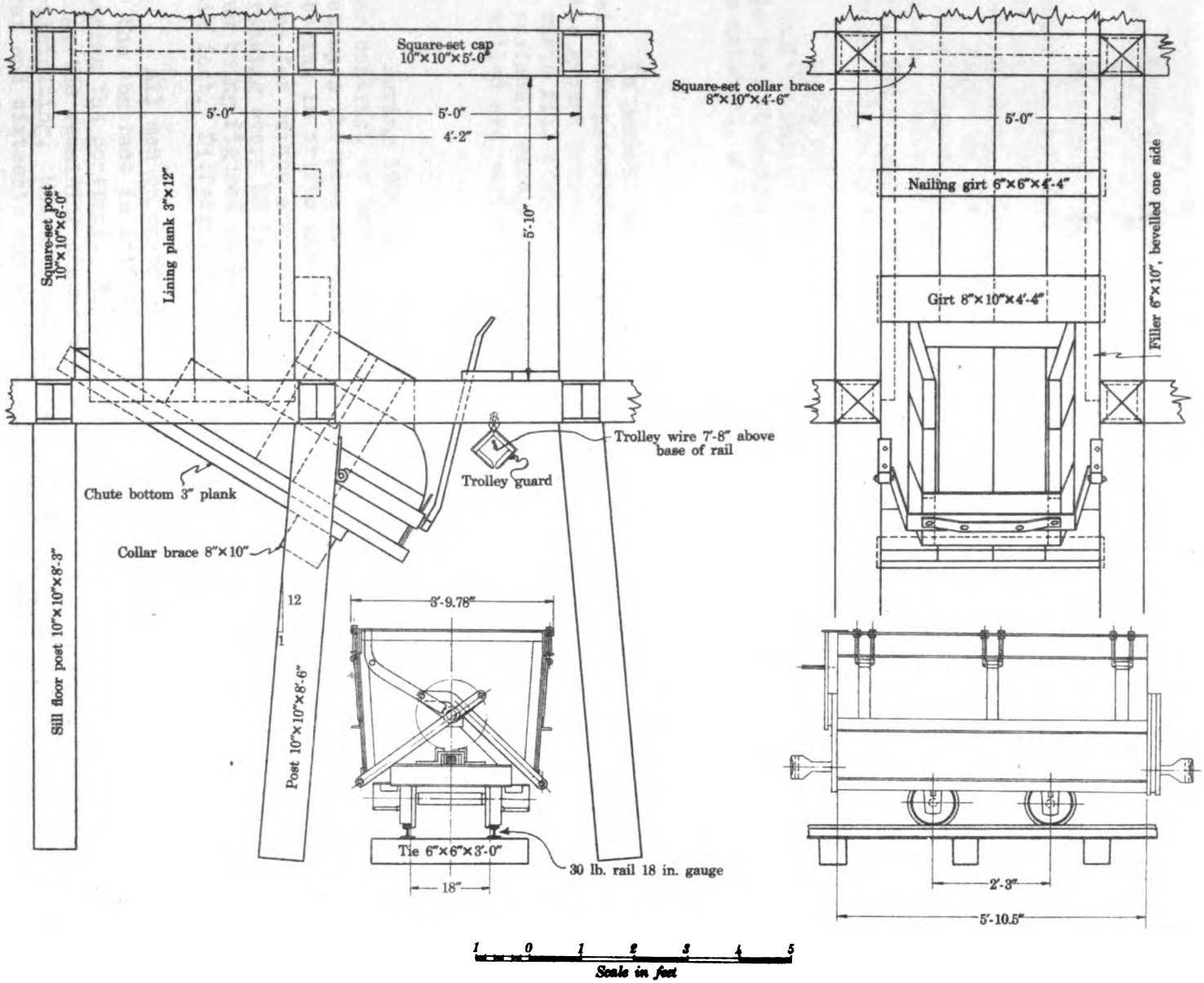

Figure 16. – Loading chute and car

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Ore Haulage to Shafts. - Ore is drawn from the stopes through chutes into cars on the main haulage levels. The chute gate is patterned after that used at the United Verde mine (see Fig. 16).

Hand-tramming with small side-dump and end-dump cars was used above the 2,000 level, which was the first to be equipped with motor haulage.

Trolley locomotives and gable-bottomed cars of 40-cubic-foot (2 tons) capacity are now used in the transportation of ore to the shafts. The locomotives are similar to but slightly smaller than that used for hauling mill ore on the 500 level. Five of them are in use. An average train consists of 10 cars. Figure 16 also shows the car, which was patterned after that used by the Miami Copper Co. The train crew consists of motorman, chute tapper, and brakeman. Under average conditions a crew will haul from 300 to 400 tons per shift. The average length of haul is about 1,000 feet. Figure 17 shows a typical haulage level.

Waste Fill Haulage. - Three 3-ton storage battery locomotives are used for hauling waste fill from the waste raises to the stopes. Eight to ten 18-cubic-foot side-dump cars comprise a train. The crew for waste fill consists of a motorman, a loader, and a brakeman.

PERCENTAGE EXTRACTED

In general more ore is mined from a given block of ground than is estimated, with practically no dilution. All stopes are filled tightly, and very little ground movement takes place. Strong gob lines or timber mats separate the waste fill from ore left in place. No waste is sorted underground or on the surface. Blocks of ground which are too low-grade as mined to be hoisted and milled are used for filling.

WAGE, CONTRACT AND BONUS SYSTEMS

During 1928 from 60 to 75 per cent of all work underground was done under contract or bonus systems. Rates per foot for drifts and crosscuts are based upon the size of opening, location, and general working conditions. All timbering in drifts and crosscuts is contracted at a price per set. Sliding scales for raises are based upon the height of the raise, the rate per foot increasing every 50 feet. Shaft sinking is done under a bonus system, the bonus being figured on ¼-foot advance. Stoping is contracted at a price per cubic foot. Different rates are paid for different types of stopes.

All men on contract are guaranteed company wages. The man responsible for the contract in drifts, crosscuts, and shaft-sinking is paid extra for managing the contract. Each man draws the regular wage for the work he is doing; the remaining amount due on the contract is divided equally among those participating. Measurements are made by the engineering department on the first and sixteenth day of the month. The contract earnings are figured by the mine superintendent and the auditing department.

Contracts are let by the mine foreman subject to the approval of the mine superintendent and general office.

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From Jan. 1, 1922, to October, 1928, the wage scale per shift at the Magma mine was as follows:

Miners - - - -	$4.95	Brakemen - -	$4.13
Timbermen - - -	5.23	Pipemen - - -	4.68
Shovelers - - -	4.13	Cagers - - -	4.68
Hand trammers -	4.13	Trackmen - -	4.95
Motormen - - -	4.68	Nippers - - -	4.13

In October, 1928, a 10 per cent increase in rates was put into effect. At this mine, as in all the important copper-mining districts in Arizona, the standard scale or base rate is paid when copper is at or under 15 cents per pound. Informal increases of 5 or 10 per cent are made at each 2 cents per pound increase in the price of copper. Corresponding decreases are made when the price of copper is reduced.

VENTILATION

The mine is mechanically ventilated by four large forward-curved-blade multivane centrifugal fans. One is on surface exhausting from the No. 4 shaft. The other three are underground as boosters on the main intake current; one is on the 2,550 level near the No. 5 shaft, and one each on the 2,000 and 2,250 levels near the No. 2 and No. 3 shafts.

The No. 4 shaft surface fan has a rotor 8 feet in diameter by 2 feet wide, and is operated by 262 r.p.m. by belt drive from a 100-hp. 2,200-volt 715 r.p.m. induction motor.

The 2,550-level booster fan is also an 8 by 2 foot fan and is operated at 285 r.p.m. by Texrope drive from a 100-hp. 2,200-volt 750 r.p.m. synchronous motor.

The 2,000 and 2,250 level boosters are 4.5 by 2.3 foot fans, direct driven at 486 and 720 r.p.m. by 75 and 100 hp. induction motors, respectively. The former ventilates a prospecting zone on the 2,000 level east only, while the latter sends No. 2 and No. 3 shafts air to the stoping zones on the 2,250 level. The ventilation system uses about 450 hp., of which the main fans take 300 hp. and the auxiliary units up to 150 hp.

The total air supply was 80,000 cubic feet per minute in February, 1929, equivalent to about 400 cubic feet per minute per man employed underground on the day shift; this quantity has since been increased to some extent. About half of the air intakes through the No. 5 shaft and the other half through the No. 2 and No. 3 shafts. All of it, except for a small amount, exhausts through the No. 4 shaft, which is being enlarged to handle a larger volume through a new fan installed - for greater accessibility - at the 500-level stations.

Except for the intake shafts, all air currents move upward, and their travel is materially assisted by large natural drafts. The natural draft, due to temperature differences between the No. 5 shaft intake and the No. 4 shaft and stope return to the 2,550 level, has been calculated as ranging from 1-in water-gage pressure in summer to 3 inches in winter.

The present mining zone is in high-temperature rock, and the ventilation system is predicated entirely on cooling the mine air to a bearable con-

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dition. Without mechanical ventilation it is doubtful whether the mine could be operated except in the immediate vicinity of the major air courses. Virgin rock temperatures range from 109 to 115° in the active zone, and yearly average surface dry-bulb temperatures are about 69° - both exceptionally high figures. Average dry-bulb temperatures in the stopes are about 88° in winter and 90° in summer, with corresponding wet-bulb temperatures about 2° lower. Practically all working places off the major air currents have auxiliary ventilation in the form of fan-tubing units using small high-speed multivane centrifugal fans and 12 and 16 inch canvas tubing. Auxiliary units on the 2,550 level in February, 1929, with a large number of stopes started, but without through ventilation, offered an extensive and intensive example of this mode of ventilation.

Air samples show that the quality of the mine air as to chemical composition is good throughout the mine.

FIRE PREVENTION

It is not thought possible to prevent smoking in the Magma mine entirely; however, it is possible to limit it to certain prescribed areas. With this in mind, the following rules are printed in the Safety Rule books of the company:

1. Smoking is prohibited on all shaft stations and in all drifts that are timbered up to a shaft.

2. Smoking is prohibited in 21 winze from the 2,550 level to the 2,000 level, and in any other manway or raise that may be used as a main air course in the future.

A copy of the Safety Rule book is given to each man when he is hired. To make sure that each new man learns this rule, it is read to him at the time he is given the book. He is also asked to be careful with his light and with cigarette stubs wherever he may work.

Each locality where smoking and open lights are prohibited is marked with a conspicuous sign:

"SMOKING AND OPEN LIGHTS PROHIBITED ON STATION"

printed both in English and Spanish. Stations are lighted by electricity, which makes carbide lights unnecessary at these places.

Lunching on shaft stations is not allowed, so that there can be no temporary accumulation of paper and scraps near a shaft.

All inflammable refusal is placed in cans and removed from the mine each shift. Separate cans are provided for spent carbide. Powder boxes are broken up and tied in bundles and with the packing sawdust and paper are taken to the surface each shift.

The nipper on each level is responsible for keeping the powder magazine on his level clean and the level clear of all inflammable refuse.

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Booster fans, wherever possible, are placed in untimbered drifts or stations. As it is not always possible to place these installations away from timber, all timbered fan stations are metal-lathed and gunited to prevent the fire from spreading in case the motor burns up.

Underground transformers are in untimbered, gunited stations, and all timber necessary near junction boxes is gunited. These stations are equipped with two 2-gallon tetrachloride extinguishers. They are hand-operated and are nonconductors of electricity. Pump stations are equipped with several small Pyrene extinguishers.

Shaft stations are equipped with a 2-inch high-pressure water line and 50 feet of hose. Fire lines are used for no other purpose except fire and are separate and distinct from all other water lines. Fire lines and hose are examined and tested once a week to make sure that they are in operating condition at all times. In addition to the water lines, each shaft station has one 2½-gallon soda-acid fire extinguisher. These extinguishers are examined and tested at regular intervals and the contents renewed as necessary.

A fire guard goes on shift one hour before each regular shift and inspects each shaft station after the men from the preceding shift have been hoisted. This is an added precaution against the possibility of starting a fire from an open light or a cigarette stub. On days when the mine is not operating, two fire guards are on shift and visit all shaft stations, motors, transformers, and other places where a fire hazard is thought to exist.

At the collar of each shaft a 4-inch water line is installed, which discharges directly into the shaft and may be used to floor it. The valves for these lines are on the surface, so that they may be reached at all times. The 4-inch lines have been installed for use in case the 2-inch fire lines and chemical extinguishers are not sufficient or can not be reached on account of smoke or heat.

The stench system is used to notify the underground workmen in case of fire. Ethyl mercaptan is turned into the compressed air line on the surface. This method has proved very satisfactory, both in speed of giving the alarm and in reaching all working places.

The ventilation current in each operating shaft is downcast. Each level, except the bottom level, has at least two sources of fresh air, thus assuring fresh air in case of fire in any one shaft. Each shaft may be cut off from the main mine workings by fire doors. These doors for the most part are hand-operated. However, they are so placed that they can be reached by an oxygen breathing apparatus crew. All doors are made of wood and are placed in rock drift or in gunited timber.

In addition to the precautions mentioned, a squad of 15 men is trained in standard mine-rescue practices and is available at all times. The company is also a member of the Globe-Miami Mine Rescue Association.

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MINE-SAFETY ORGANIZATION OF THE MAGMA COPPER CO.

The safety work of the Magma Copper Co. is in charge of a safety engineer who is responsible to the general manager.

The mine safety organization consists of workmen's committees who make regular safety inspections of all working places in company with the mine foreman and safety engineer. The recommendations of these committees are submitted to the mine superintendent for approval.

An underground safety meeting is held once a month on each level of the mine. These meetings are conducted by the safety engineer, who talks on various phases of safety and also receives safety suggestions directly from the workmen. The suggestions are reported to the mine foreman and mine superintendent for action.

A bosses' safety meeting is held once a month, at which accidents, accident prevention, and standard mining practice are discussed.

All bosses receive a bonus for accident prevention, based upon 1,000 shifts worked.

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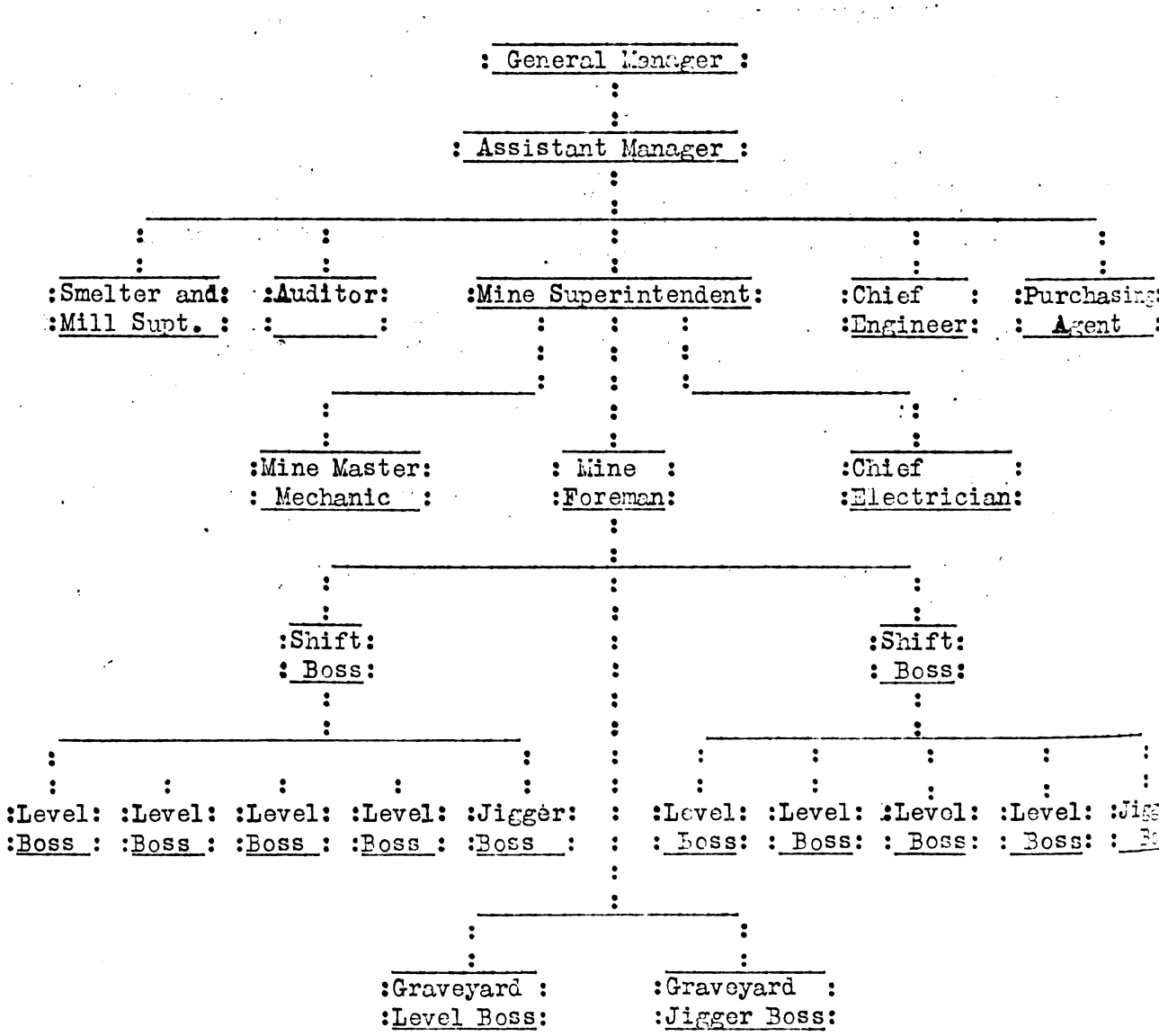

Figure 18. - Chart of administration organization.

There are about 500 men in the mine, supervised as follows: 250 men under each shift boss, 30 to 40 men under each level boss, and 15 to 20 men under each jigger boss. The shift boss is directly in charge of the mine, and might be considered a swing foreman in other mines. The level boss is in charge of all work on a level or part of a level. Jigger bosses are used on special jobs in comparatively isolated sections of the mine, such as waste fill, and also on haulage. Figure 18 is a chart showing the adminstration organization.

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MINING COSTS

Table 1. - Summary of Costs

Name of mine: Magma mine. Period covered: Year of 1928.

Tons hoisted during period: 263,094.

Mining methods: Timbered rill, rill and pillar.

Underground Costs per Ton of Ore Hoisted

	1 Labor	2 Supervision	3 Comp.air Drills and Steel	4 Power	5 Explosives	6 Timber	7 Other Supplies	8 Total
Development -	.502	.044	.046	.023	.059	.062	(1)	.736
Mining - - - -	1.490	.267	.280	.134	.150	.470	.058	2.849
Transportation	.484	-	-	.040	-	-	(2) .092	.616
General expense	.741	-	-	-	-	-	(2) .425	1.166
Surface expense directly applicable to underground operation - -	.126	-	-	-	-	-	.004	.130
Total	3.343	.311	.326	.197	.209	.532	.579	5.497

(1) Included under mining. (2) Includes some new equipment.

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Table 2. - Summary of costs in units of labor, power, and supplies

Magma Mine, Year 1928.

Tons of Ore Mined and Hoisted: 263,094.

Mining Methods - Timbered Rill, Rill and Pillar

A. Labor (man-hours per ton hoisted):

Breaking, timbering and mucking:	
(1) Development	0.502
(2) Mining, including filling	1.706
Haulage and Hoisting	0.639
Supervision	0.479
General	0.934
Total labor underground	4.260
Average tons per man shift underground	1.880
Average tons per man shift on surface properly chargeable to underground	(1) 6.481
Average tons per man shift, including surface	1.674
Labor, percentage of total cost	55.656

B. Power and Supplies:

Explosives (lbs. per ton)	
Development	0.2
Mining	0.7
Timber (bd.ft. per ton)	
Development	2.0
Mining	(2) 17.0
Total power (kw. h. per ton)	34.612
Air compression	14.165
Hoisting and tramming	8.917
Pumping	5.536
Ventilation	4.914
Lighting	.263
Miscellaneous	.817
Other supplies, in percentage of total supplies and power	(3) 47.75
Supplies and power, percentage of total cost	31.63

(1) Not included in Total Labor Underground.
(2) Includes maintenance.
(3) Includes some new equipment.

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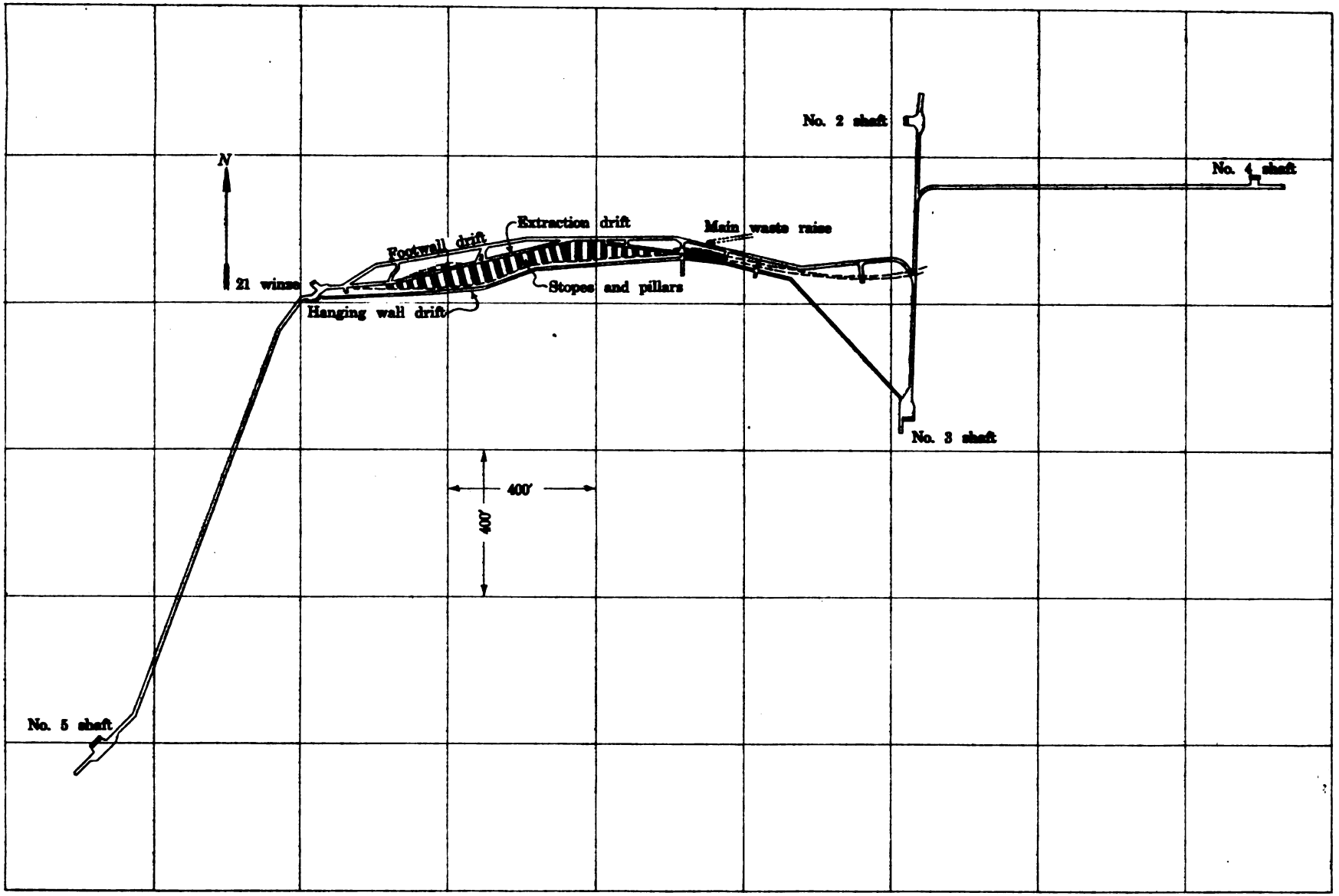

Figure 17. – Typical haulage level

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